MILLING AND PROCESSING TUNGSTEN

BY J. B. ZADRA

* * * * * * * * * * **Information Circular 7912**

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MILLING AND PROCESSING TUNGSTEN[1/]

by

J. B. Zadra[2/]

INTRODUCTION AND SUMMARY

Tungsten, a comparatively new metal, is one of the important strategic war materials. In peacetime, tungsten is widely used as a filament in electric lamps and electronic tubes and is a vital constituent of some of the most important industrial alloys. High-speed tool steel containing tungsten and stellite alloys containing cobalt, chromium, and tungsten are extremely hard and resistant to corrosion. These alloys and the tungsten carbides are unequaled as cutting tools and in other wear-resistant applications. They play a prominent role in the present era of mass production. Alloys containing tungsten resist oxidation at high temperatures and have been key alloys in the development of modern aircraft. The adaptability of tungsten to new uses is expanding, particularly in the high-temperature range.

Domestic production of tungsten has come from a large number of deposits in 17 of the 49 States; but, of the total domestic output from 1900 to 1956, more than 80 percent was produced from mines in North Carolina, California, Colorado, and Nevada. The average tenor of the mined ore was less than 1 percent tungsten trioxide (WO_3).

Various milling methods are used for concentration, including gravity, flotation, magnetic separation, and a combination of these methods, to produce concentrates that contain from 50 to more than 70 percent WO_3. Slimes that are difficult to treat by any of the above methods are usually processed chemically, with lower grade material, to produce synthetic scheelite containing 70 to 72 percent WO_3.

Although some domestic tungsten mines contain large reserves, the industry faces economic instability because of the low-grade nature of the deposits, high production costs, and severe fluctuations in consumption and price. Economic instability in the past was partly caused by unpredictable military requirements. Between 1952 and 1956 the industry operated under temporary support provided by the domestic tungsten-purchase program. It has since

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1/ Work on manuscript completed February 1959.

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been required to compete with foreign producers at world market prices that are less than the cost of domestic output. Higher, more stable prices for the metal, search for new deposits, and improved mining and milling methods to reduce processing costs are needs of the industry.

The purpose of this report is to present the technologic status and the problems encountered in milling and processing tungsten ores. The report summarizes milling practices and chemical treatment methods used at most of the major and at a number of smaller domestic tungsten operations and provides flowsheets developed for them. It also discusses production and uses of tungsten metal and analytical methods.

HISTORY

Discovery and Early Research

The word "tungsten" is derived from the Swedish words "tung" and "sten," meaning "heavy stone." In 1758 A. F. Cronstedt applied the word to the mineral scheelite because of its high density.

The origin of the word "wolfram" is obscure. It is believed that the word stems from the German words "Wolf" and "Ram" or "Rahm" (froth), which literally mean the foam of the wolf and suggests that the ore has wolflike properties. This concept resulted from the tendency of wolframite, which is frequently associated with tin ore and originally believed to be a mineral of tin, to "eat up" tin during smelting operations. It was not realized at the time that the detrimental impurity was a new element. In 1781 Carl Scheele, a Swedish chemist, demonstrated that tungsten ore (scheelite) contained peculiar acid, which was combined with lime and which he called tungstic acid. In the same year Torbern Bergman believed that the base of the acid was a metal, but he was unable to isolate it.

In 1783 the de Elhuyar brothers, Spanish chemists, showed that the mineral wolframite contained the same acid discovered by Scheele. They succeeded in producing small globules of elemental metal, which they named "wolfram," by heating a mixture of tungstic acid and charcoal in a crucible. They also showed that the metallic element was associated with iron and manganese instead of lime.

Early Production and Use

In 1847 patents were issued to Oxland in England for manufacturing sodium tungstate, tungstic acid, and metallic tungsten from wolfram-tin ores. These were followed by patents issued Oxland and Mushet in 1857 for the production of certain alloys with iron, steel, and other elements.

The first important commercial use of tungsten was in 1868, when Mushet produced a tungsten-manganese air-hardening steel. In 1898 F. W. Taylor and Maunsel White developed the composition known today as high-speed steel. The Taylor-White steels, manufactured by the Bethlehem Steel Co., were exhibited at the Paris exposition in 1900 and were received with great enthusiasm.

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Their ability to hold a cutting edge at dull red heat promised to revolutionize the tool-steel industry.

Development of Tungsten Uses Since 1900

Most commercial development and growth of tungsten uses have taken place since 1900. During this time an important application of tungsten has been in electric-lamp filaments. At first, growth of this use was slow because of the difficulty of producing a ductile metal from tungsten-metal powder. In 1904 Just and Hannaman achieved success in making filaments by extruding a mixture of fine tungsten powder and an organic binder through proper-size dies and volatilizing the binder in hydrogen to prevent oxidation. The tungsten particles were then sintered to form a conducting filament suitable for lamps, but the product lacked ductility. In 1909 Coolidge obtained his first patent for producing ductile tungsten by heat treatment and mechanical working, and by 1911 incandescent lamps using coiled tungsten filaments were manufactured commercially.

Tungsten carbide, first produced by H. Moissan in 1896 by reducing tungstic trioxide with carbon, was used successfully as early as 1919 in manufacturing tools and dies. However, the problem of brittleness was not fully solved until 1927, when tungsten carbide was bonded with metallic cobalt at the Krupp Laboratory at Essen, Germany, and the cemented tungsten carbide was used in cutting tools, wire, drawing dies, and other items. Other carbides, such as titanium and tantalum, were used later to form mixed cemented carbides for cutting operations and special purposes. These products were made by pressing the mixtures of metallic carbide powders into desired shapes and sintered at high temperatures.

Although aluminothermic, silicothermic, and crucible methods of manufacture were used in earlier years, ferrotungsten, as an additive to steel, was first produced in the United States between 1916 and 1918. It was made in small, knockdown, fire-brick electric furnaces using 4-inch graphite electrodes. Buttons weighing 300 to 400 pounds were produced, with an approximate yield of 70 percent. In 1929-30, a new electric reduction furnace was designed that produced a button of ferrotungsten weighing 14,000 pounds, with a yield above 90 percent. This type of furnace is still being used today to produce ferrotungsten.

Current tungsten research is directed toward development of alloys for use at 1,800° to 2,500° F. or higher. The basis for this type of research is indicated by the following notes from an address by R. H. Thielemann (43).3/

The compositions of six alloys that are well known and have been used in substantial quantities in critical, high-stressed, high-temperature applications are listed below:

3/ Underlined numbers in parentheses refer to items in the bibliography at the end of this report. Pages cited refer to the item and not to this publication.

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Important Tungsten-Containing, High-Temperature Alloys

	TPA valve steel	Cyclops 17-W	S-816	NA-22H (Blaw-Knox)	X-40 (Stellite 31)	L-605 (Stellite 25)
Carbon....	0.50	0.50	0.50	0.50	0.50	0.15 max.
Chromium..	12.0	15.0	20.0	28.0	25.0	20.0
Nickel....	12.0	15.0	20.0	48.0	10.0	10.0
Molybdenum	-	-	4.0	-	-	-
Columbium.	-	-	4.0	-	-	-
Tungsten..	3.0	4.0	4.0	5.0	7.5	15.0
Cobalt....	-	-	45.0	-	Balance.	Balance.
Iron.....	Balance.	Balance.	5.0 max.	Balance.	2.0 max.	2.0 max.
Form.....	Wrought	Wrought.	Wrought.	Cast.	Cast.	Wrought.
Principal use.....	Exhaust valves in aircraft engines.	Turbosupercharger blades and disks.	Jet-engine blades.	Furnace supports.	Jet-engine blades.	Combustor components in jet-engines.

It is significant that tungsten is present in all six of these alloys. The TPA valve steel developed by the French in World War II was, until recently, used almost universally for exhaust valves in aircraft and other high-performance engines. The Cyclops 17W alloy was a modification of the TPA valve steel and was used for the turbine blades and disks in the early General Electric turbosuperchargers. The S-816 alloy was developed for the jet-engine blades in J-33, J-35, and J-47 jet engines and is still used in these applications. The cast X-40 (Stellite 31) alloy was developed in 1943 and is still being used for turbine blades in our most advanced jet engines. The Blaw-Knox alloy (NA-22H) was developed as a support material for boiler tubes and furnace components having strength, oxidation resistance, and stability over long periods at temperatures up to 1,800° F. The L-605 alloy (Stellite 25) is being used extensively for combustion components in jet engines, rocket motors, and other applications where temperatures may reach 2,000° F. or higher. Many other alloys, such as the 18-4-1 high-speed tool steel, that have performed well in many critical applications, depend upon the properties imparted by tungsten.

Tungsten Resources, Production, Consumption, and Price

The commercial tungsten minerals are wolframite, ferberite, hubnerite, and scheelite. Wolframite (including hubnerite and ferberite) is the most important world source of tungsten, whereas scheelite is the principal domestic source. In recent years scheelite has comprised about 75 percent of the United States tungsten production and wolframite the remaining tungsten. Total world reserves of tungstic trioxide are estimated to be about 175 million short-ton units.[4/] Asia has the largest and richest deposits, although North America, South America, Europe, and Oceania have low-grade deposits.

4/ A short-ton unit is 20 pounds of WO_3. Ore assaying 1 percent WO_3 contains one unit per ton.

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Since World War I the United States has depended on foreign imports for much of its tungsten. The largest supply came from China, which supplied about 40 percent of the domestic requirements until 1939, when the Chinese source of supply was cut off, causing a tungsten shortage in the United States. This factor, plus the outbreak of war late in 1939, brought about an increased domestic demand for tungsten.

To stimulate domestic tungsten production to meet the war demand, the Metals Reserve Company, in February 1942, established a Government stockpiling program, with purchasing depots in California, Nevada, Arizona, New Mexico, and Colorado. Prices paid from April 1942 to December 1943 to small producers were $24 per short-ton unit of Specification-grade (60 percent WO_3) concentrate. The program was later extended to December 1944 and the price increased to $30 per unit. On April 30, 1944, the Metals Reserve Company discontinued its premium-price payment, which resulted in the closing of several mines and decreased production in others.

In 1945, following World War II, imports and consumption decreased. In the latter part of the year, however, industrial activities increased the demand for tungsten, and in July 1946 Congress amended the Stockpiling Act for the acquisition and retention of stocks to eliminate risks in emergencies. This action, combined with the increased domestic demand for tungsten, led to an increase in the price of imported tungsten to $33 per short-ton unit of WO_3 and $32 per short-ton unit of WO_3 in domestic scheelite. In spite of the increased price and the demand for tungsten, domestic production decreased. Production was 4.6 million pounds in 1946 and decreased 65.2 percent to 3 million pounds in 1947; however, imports remained at the same level.

Expanded industrial activities in 1948 again increased the demand for tungsten. During the year domestic production reached 4 million pounds, an increase of 33 percent over 1947. Imports increased 25 percent to 7.5 million pounds, and consumption increased to 8.8 million pounds compared with 7.8 million pounds in 1947.

A decrease in industrial activities in 1949 caused prices and domestic production to decline. Import prices dropped to a low of $18 per short-ton unit of WO_3. Domestic output decreased to 2.9 million pounds, imports to 6.2 million pounds, and consumption to 4.9 million pounds.

The hostilities in Korea early in 1950 created a demand for tungsten, which with the loss of supply from China and Korea, led to a rapid increase in the price of tungsten. In 1951 demands exceeded the combined domestic production and imports, and the net result was depletion of inventories. To overcome this shortage, the Government in April 1951, instituted drastic restrictions in the use of tungsten and at the same time provided a long-term purchase program designed to stimulate domestic production. The program provided for the purchase of 3 million short-ton units over a 5-year period or to July 1, 1956. The price for domestically mined tungsten was set at $65 per unit. The program was later extended to July 1, 1958, but the price was lowered to $55 per short-ton unit; however, failure of the Congress to provide adequate funds

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caused suspension of purchases in December 1956. The incentive established in 1951 stimulated domestic production. The 1953 output of 9.2 million pounds exceeded consumption. In 1954, 1955, and 1956, production of 13.1, 15.8, and 14.3 million pounds, respectively, greatly exceeded consumption.

Figure 1 shows the trends in domestic ore produced, consumption, and selling price from 1900 to 1956, figure 2 the location of tungsten mills in California, figure 3 the location of tungsten mills in Nevada, and figure 4 the tungsten-bearing zones in the Western States.

TUNGSTEN MINERALS

Tungsten never occurs free in nature and is found only in a few minerals. Of the known tungsten-bearing minerals, the four of commercial importance are: Ferberite, $FeWO_4$; wolframite, (Fe, Mn) WO_4; hübnerite, $MnWO_4$; and scheelite, $CaWO_4$. Hess and Schaller (26) divide these minerals into two groups (wolframite and scheelite); the first group contains ferberite, wolframite, and hübnerite; the second contains scheelite.

The economic tungsten minerals are tungstates, in which the tungsten is combined with oxides of calcium, iron, and manganese. Although a number of

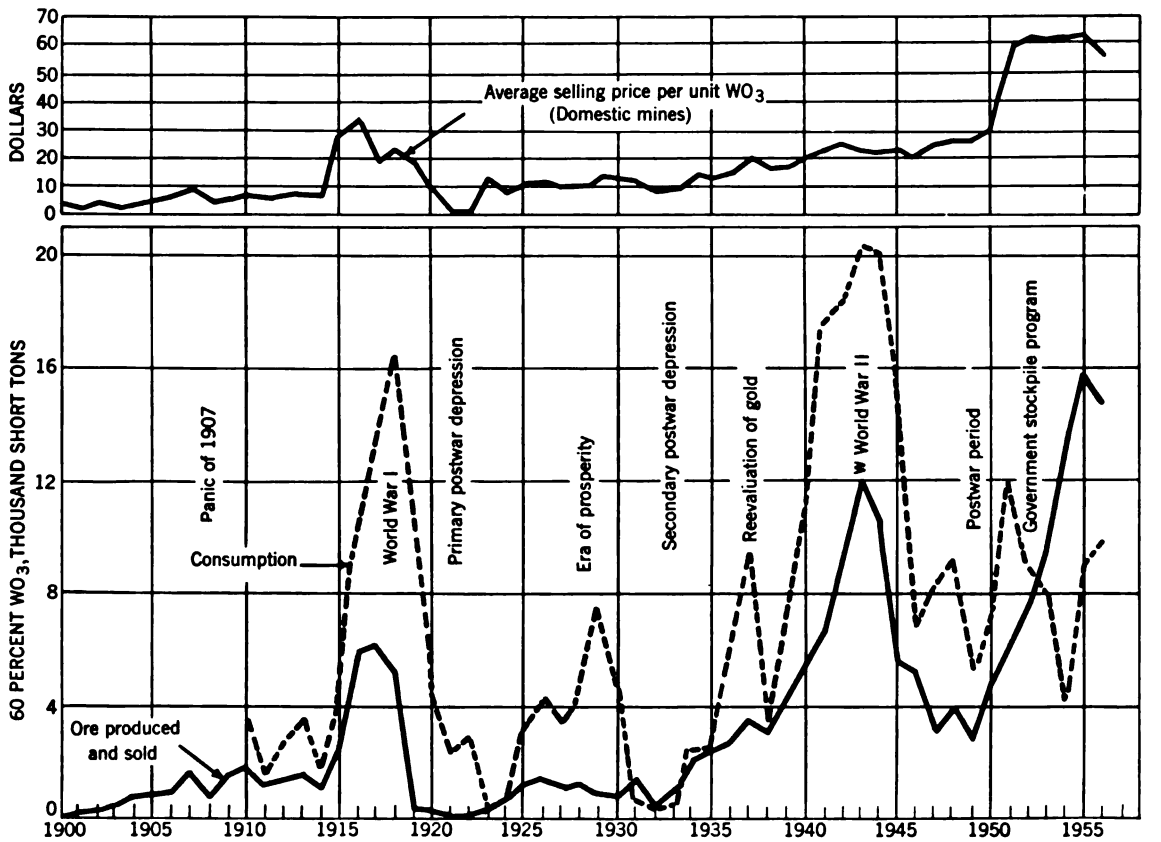

FIGURE 1. - Trends in Domestic Tungsten Ore Produced, Consumption, and Selling Price, 1900-56.

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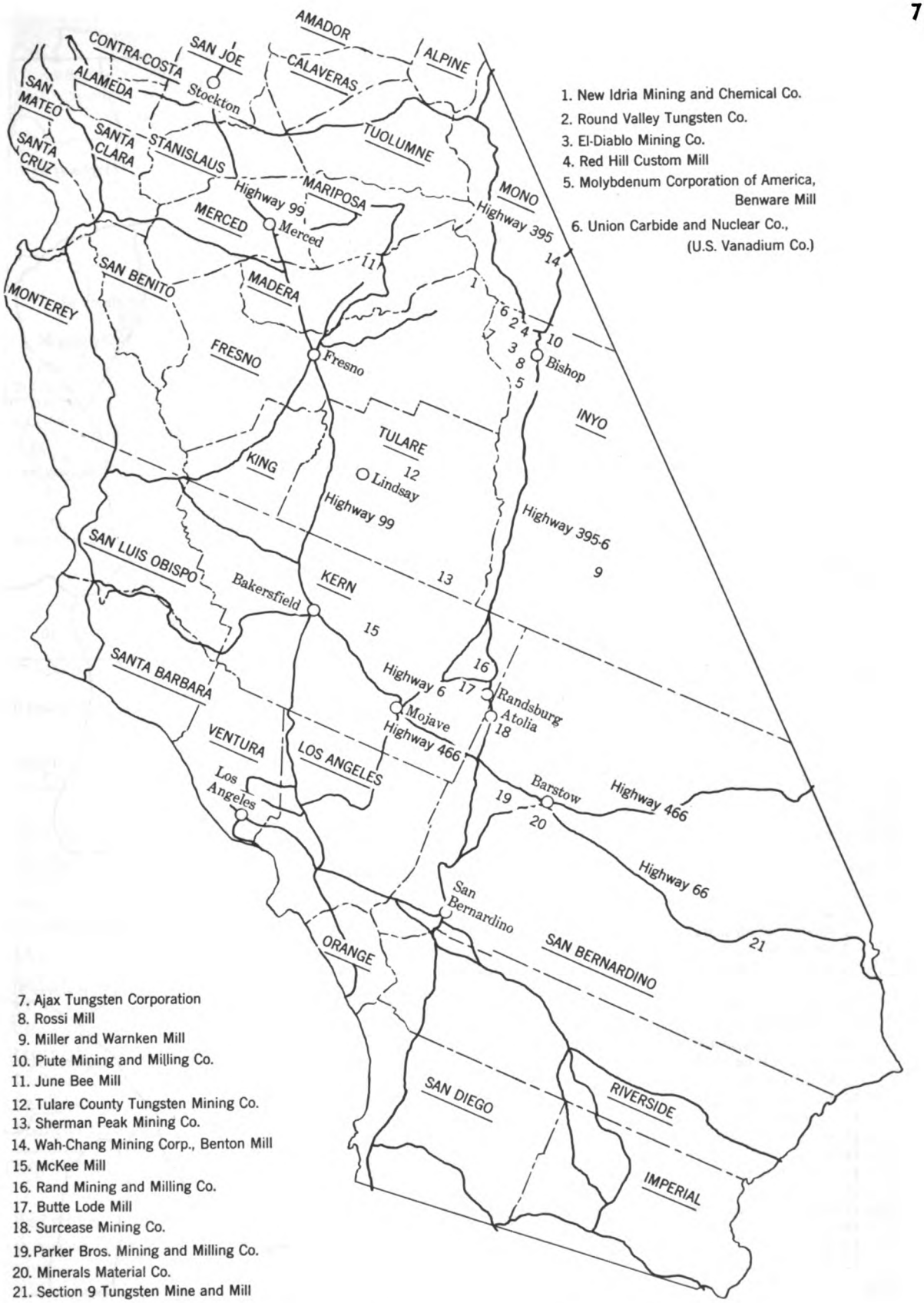

FIGURE 2. - Location of Tungsten Mills in California.

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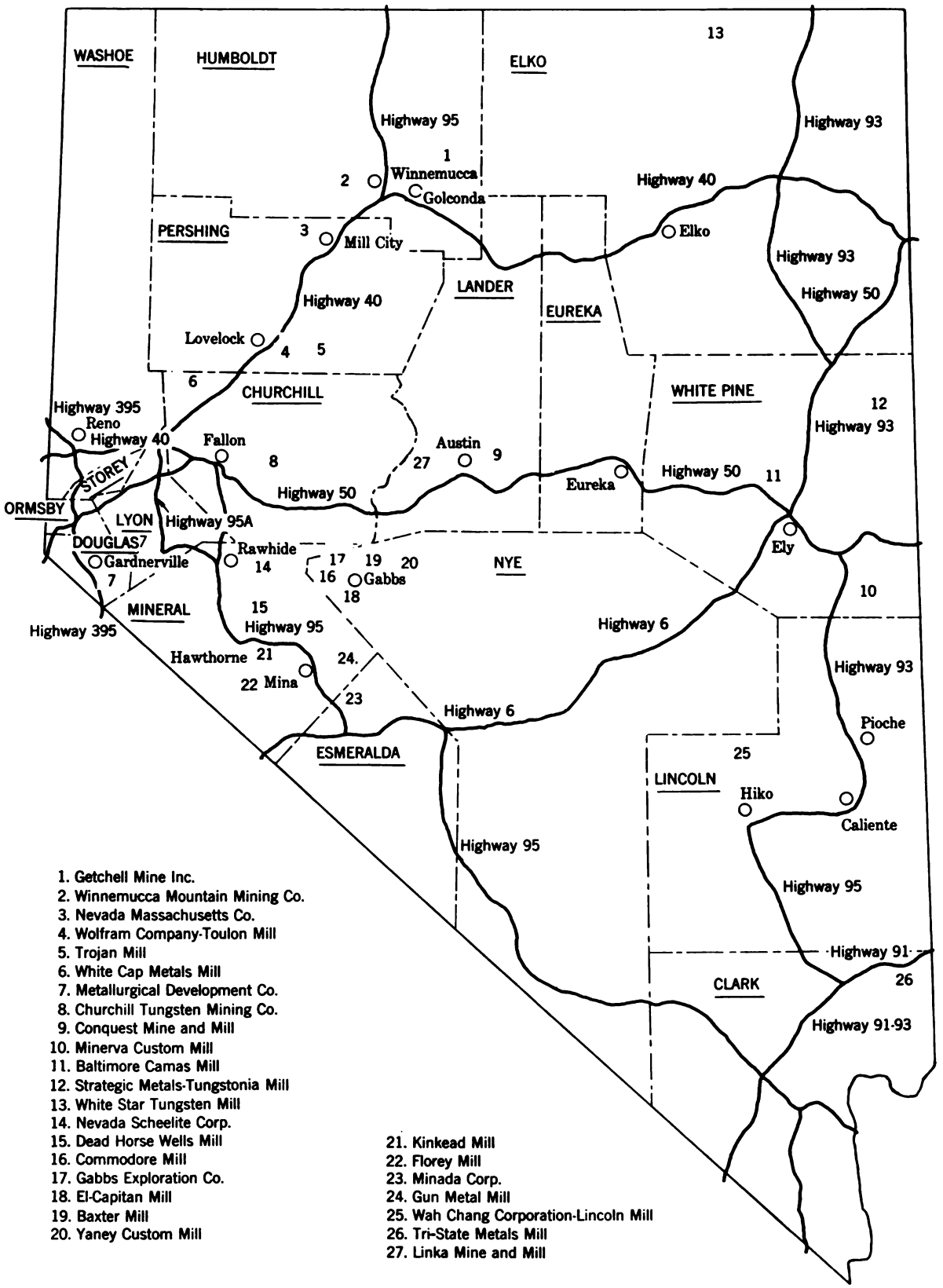

FIGURE 3. - Location of Tungsten Mills in Nevada.

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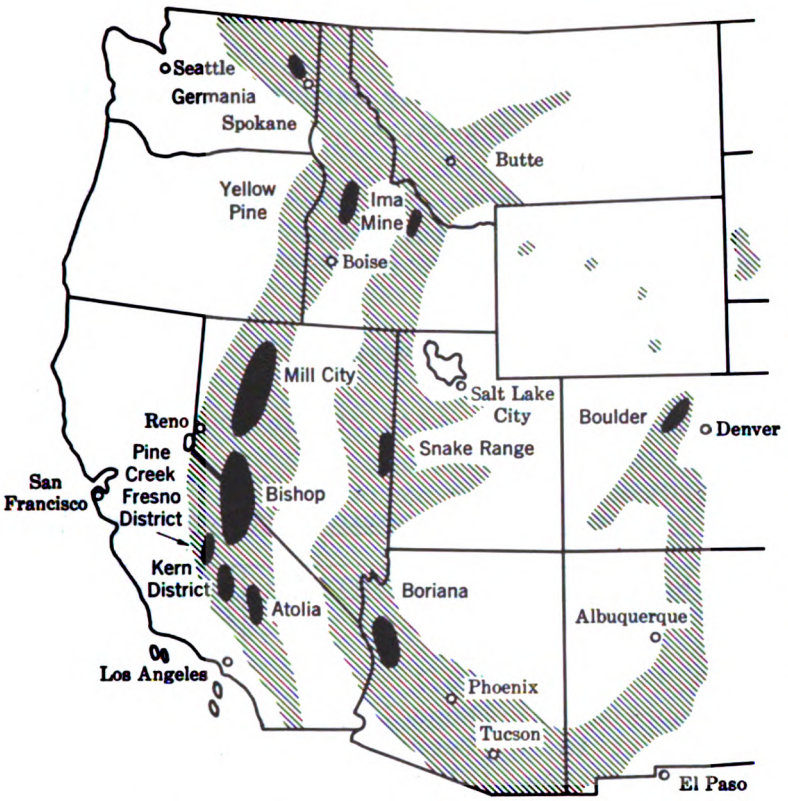

FIGURE 4. - Tungsten-Bearing Zones in the Western States.

tungstic acids have been prepared artificially, salts of only one acid have been found in nature, namely, those of the normal tungstic acid (H_2WO_4). Tungsten has been identified in a number of minerals as an impurity, but only 12 distinct species of minerals containing tungsten are known. In virtually all of its minerals, tungsten is one of the chief constituents; the two exceptions are powellite and chillagite. Table 1 lists the known tungsten minerals and gives their theoretical composition and the percentage of tungsten contained in each mineral.

Stolzite and raspite differ only in crystal form. Stolzite is tetragonal, and raspite is monoclinic.

TABLE 1. - Tungsten minerals

Mineral	Composition	Tungsten, percent 1/
Scheelite.....	Calcium tungstate ($CaWO_4$)	63.9
Ferberite.....	Iron tungstate ($FeWO_4$)	60.6
Wolframite.....	Iron-manganese tungstate ($(Fe,Mn)WO_4$)	60.6-60.7
Hübnerite.....	Manganese tungstate ($MnWO_4$)	60.7
Powellite.....	Calcium-tungstomolybdate ($(Ca,Mo)WO_4$)	-
Chillagite.....	Lead tungstate-lead molybdate ($3PbWO_4.5PbMO_4$)	17.2
Stolzite.....	Lead tungstate ($PbWO_4$)	40.4
Raspite.....	do.	40.4
Cuprotungstite...	Hydrous copper tungstate ($CuWO_4.2H_2O$)	52.9
Tungstite.....	Hydrous tungsten trioxide ($WO_3.H_2O$)	73.6
Tungstenite.....	Tungsten sulfide (WS_2)	74.6
Ferritungstite...	Hydrous iron-tungsten oxide ($Fe_2O_3.WO_3.6H_2O$)	41.5

1/ Theoretical.

Reinite ($FeWO_4$), as described by Dana (12, p. 991), may be regarded as nearly pure ferberite, pseudomorphic after scheelite.

According to Dana, cuproscheelite is a tungstate of copper and calcium (Ca, Cu)WO_4; it may be scheelite partly altered to cuprotungstite, in which the calcium has not been wholly replaced by copper.

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Meymacite, another alteration product described by Carnot, may have the formula $WO_3.H_2O$ and is probably the same as tungstite.

Dana's System of Mineralogy also mentions the small quantities of tungsten are contained in the columbium-tantalum minerals, but the information concerning them is so incomplete that their existence as definite mineral species cannot be considered authentic.

PROPERTIES OF ECONOMIC TUNGSTEN MINERALS

Table 2 gives the physical and chemical properties of the four principal tungsten minerals.

TABLE 2. - Physical and chemical properties of economic tungsten minerals

Mineral	Ferberite	Wolframite	Hübnerite	Scheelite
Composition (pure)	$FeWO_4$	$(Fe,Mn)WO_4$	$MnWO_4$	$CaWO_4$
WO_3, percent.....	76.3	76.5	76.6	80.6
W, percent.....	60.6	51.3	60.7	63.9
Crystallization...	Monoclinic	Monoclinic	Monoclinic	Tetragonal
Cleavage.....	Perfect in 1 direction	Perfect in 1 direction	Perfect in 1 direction	Good in 4 directions
Specific gravity..	7.5	7.1-7.5	7.2-7.3	5.4-6.1
Color.....	Black	Dark gray to black	Reddish brown to black	Pale yellow brown, commonly white
Tenacity.....	Very brittle	Very brittle	Very brittle	Very brittle
Luster.....	Submetallic to metallic	Submetallic to metallic	Submetallic to adamantine	Vitreous to resinous
Fracture.....	Uneven	Uneven	Uneven	Uneven
Hardness.....	5	5-5.5	5	4.5-5
Magnetism.....	Sometimes feebly magnetic	Slightly magnetic		Nonmagnetic
Streak.....	Dark brown to nearly black	Dark brown to black	Brownish red greenish yellow	White
Diaphaneity.....	Opaque to translucent in cleavage plates	Opaque	Opaque to translucent	Transparent to translucent
Common form of occurrence.....	Well-defined crystals, massive cryptocrystalline	Irregular masses	Radiating groups of thick-bladed crystals	Massive and small grains

Ferberite, wolframite, and hübnerite form a continuous series of iron-manganese tungstates, of which ferberite is a member at the iron end and hübnerite a member at the manganese end. The iron and manganese contents of the series are so variable that mineral identity cannot be distinguished

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without chemical analysis; therefore, the entire series is often called wolframite. No specimen of ferberite free from manganese or hübnerite free from iron has been found in nature. Since the wolframites form a complete series with an indefinite number of members, any classification of the minerals into ferberite, wolframite, and hübnerite must be arbitrary. The following definitions, proposed by Hess and Schaller (26), are generally accepted.

Ferberite. - A monoclinic iron tungstate having, when pure, the composition $FeWO_4$. It may contain not more than 20 percent of the hübnerite molecule $MnWO_4$.

Wolframite. - A monoclinic mineral containing the ferberite molecule ($FeWO_4$) and the hübnerite molecule ($MnWO_4$) in all proportions between 20 percent $FeWO_4$ and 80 percent $MnWO_4$ and 80 percent $FeWO_4$ and 20 percent $MnWO_4$.

Hübnerite. - A monoclinic manganese tungstate having, when pure, the composition $MnWO_4$. It may contain not more than 20 percent of the ferberite molecule $FeWO_4$.

The wolframites are metallic in appearance, and scheelite is vitreous or resinous in luster.

The economic tungsten minerals are characterized by high specific gravities ranging from 5.4 for scheelite to 7.5 wolframite (table 2). The tungsten minerals are two to three times as heavy as quartz, which has a specific gravity of 2.6. This wide difference in specific gravity is utilized in separating quartz, calcite, and other lower specific gravity minerals when ores are concentrated by gravity methods.

Because of their similarity in chemical composition and the fact that the atomic weights of iron and manganese are almost the same (Fe 55.84; Mn 54.93), the wolframites contain approximately the same percentage of tungsten trioxide.

Each of the minerals in the wolframite series has a perfect plane of parting (cleavage) in one direction. Scheelite has eight planes of parting; four are pronounced, and four are not.

The principal tungsten minerals have a hardness of about 5 on the Mohs scale of hardness, which is: 1, Talc; 2, gypsum; 3, calcite; 4, fluorite; 5, apatite; 6, feldspar; 7, quartz; 8, topaz; 9, sapphire; and 10, diamond. The tungsten minerals are easily scratched with the blade of a good pocketknife, which has a hardness of about 5-1/2.

The streak of a mineral (the color of its fine powder) is best obtained by rubbing the mineral on a piece of unglazed porcelain and brushing off the excess. It may be obtained less satisfactorily by pulverizing a fragment of the mineral or by scratching the mineral with a pocketknife or file. The streaks of the wolframites range from black for the wolframites through brown to brownish red for the hübnerite. The streak of scheelite is white.

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MINERALS ASSOCIATED WITH TUNGSTEN ORES

The tungsten ores of the United States occur in six types of deposits - segregation, pegmatite, replacement, contact-metamorphic, veins, and placers. The tungsten minerals (26, pp. 37-45) in these deposits are hübnerite, ferberite, wolframite, and scheelite. Contact-metamorphic deposits containing tungsten invariably contain scheelite and are the greatest source of domestic production. Domestic contact-metamorphic deposits are found in the Great Basin region of California, Nevada, western Utah, Montana, Arizona, New Mexico, Oregon, and Washington. They are formed in intruded rocks of calcareous composition by solutions and gases that emanate from the intruded magma (46, p. 4).

The original calcareous composition of the carbonate material is metamorphosed during intrusion to an aggregate of contact-metamorphic minerals, garnet, epidote, calcite, diopside, actinolite, scheelite, zoisite, vesuvianite, clinozoisite, and quartz. Potash and magnesium micas, pyrite, molybdenite, zeolites, and many other minerals are present in some contact-metamorphic deposits (25, p. 249).

The tungsten mineral of the Nederland and Boulder districts in Boulder County, Colo., is ferberite, which occurs in fine-grained quartz called "horn rock" (34) in Cambrian gneissic quartz monazite. The minor constituents of the veins include hematite, magnetite, fluorite, dickite, ankerite, siderite, barite, adularia, montmorillonite, halloysite, beidellite, scheelite, and opal.

The Climax molybdenum deposit (7), one of the world's largest in Lake County, Colo., contains the tungsten mineral hübnerite, which is recovered as a byproduct concentrate from the molybdenum ores. The primary minerals in the deposit are chalcopyrite, fluorite, hübnerite, molybdenite, orthoclase, pyrite, quartz, sericite, sphalerite, and topaz. The secondary minerals are chalcocite, jarosite, limonite, and molybdite. Hübnerite occurs with sericite in small quartz veins that cut the silicified rock and the molybdenite veins. It is also present in many quartz-pyrite veins that cut all zones of alteration including the molybdenite veins.

The tungsten mineral of the Ima-mine ore body in Lemhi County, Idaho (17), is hübnerite that occurs in quartz veins. The veins are present in quartzite near its contact with granite intrusives, and some ore also is found within the granite. The principal metal-bearing minerals in the veins are hübnerite, silver-bearing tetrahedrite, galena, chalcopyrite, sphalerite, and molybdenite. The principal gangue minerals are fluorite, rhodochrosite, mica, talc, and pyrite.

The tungsten minerals in the ore of the Tungsten Mining Corp., Vance County, N. C. (18), are hübnerite and scheelite that occur in quartz veins. The principal tungsten mineral, hübnerite, is accompanied by minor amounts of scheelite. Quartz is the dominant gangue mineral, with subordinate amounts of fluorite, rhodochrosite, and sericite. Pyrite, the predominant metallic mineral is accompanied by some chalcopyrite, galena, sphalerite, and tetrahedrite; molybdenite is extremely rare.

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ORE-DRESSING TUNGSTEN ORES

Since World War I the art of beneficiating tungsten ores has made great strides. As late as 1916, grinding equipment consisted largely of gravity stamps. Their use resulted in high losses of tungsten through production of excessive slimes. Stage crushing or size classification had not yet been well developed. The ore were classified by screening and gravity concentration, using shaking tables, vanners, and canvas-covered tables. In the light of present knowledge, it is not surprising that the average recovery at these early mills was as low as 60 percent or less.

In recent decades exploitation of contact deposits in California and Nevada has given rise to certain milling problems that have resulted in the use of flotation as an adjunct to gravity concentration in treating scheelite ores. The comparatively high losses due to sliming of scheelite during milling have commanded the attention of several investigators. Although flotation has increased the overall recovery of tungsten minerals and despite advances in the art over the past 30 years (3, 19, 31, 42), achieving high recovery of the tungsten mineral from slimes still requires a continued, intensive research effort. However, the recovery of tungsten from slimes and low-grade products by chemical means to produce synthetic scheelite has proved satisfactory.

Slimes and Their Characteristics

In discussing ore-dressing methods for tungsten ores and milling practices at principal mines, the words "slime" or "slimes" are used frequently. For clarity it is appropriate to discuss the constitution of a slime and its characteristics.

The mineral scheelite is friable, and in crushing and grinding the ore is more readily reduced to fine particles than the associated gangue minerals. In grinding the ore finely enough to liberate most of the scheelite, a substantial portion (5 to 20 percent) is inadvertently reduced to a finer size than that necessary to liberate the scheelite mineral. This portion is usually refractory to flotation. Although flotation is the method used to recover minerals in fine sizes, it has limitations as to particle size. Definite reduction in floatability, with reduction in particle size beyond a critical size of maximum floatability, is now an established fact. The flotation of fine sizes of nonmetallic and nonsulfide minerals usually begins to fail on sizes smaller than 20 microns. Moreover, these extremely fine mineral particles become slime-coated, rendering flotation reagents ineffective for their recovery. For all practical purposes, then, a true slime can be said to be a mineral particle smaller than 20 microns.

Unfortunately, in tungsten-milling operations the word "slime" is used rather loosely. For example, in a tungsten-milling operation that utilizes gravity and flotation methods it is customary to size the ground product into several fractions or into coarse, medium, and fine granular products suitable for table concentration using cones, hydraulic classifiers, or equivalent equipment. Mineral particles, although granular, but too small to respond to table concentration, come over in the cone or hydraulic classifier overflow

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and are termed "slimes" rather than "flotation feed." A practical mill slime therefore is something different from a true slime. It contains granular mineral particles (larger than 20 microns) that respond to flotation treatment and also particles (less than 20 microns) that do not respond to flotation. This condition explains the difficulty in achieving high recovery of tungsten minerals from slimes or production of high-grade concentrate.

Conditions Governing Ore Dressing

The ore-dressing or milling methods selected for beneficiating tungsten ores depend largely upon the nature and characteristics of the tungsten minerals present. A flowsheet for tungsten ores free of other heavy minerals is relatively simple; refractory or complex ores, being more difficult to treat, require ore-dressing studies to develop applicable methods. The presence of such minerals as pyrite, arsenopyrite, pyrrhotite, sphalerite, calcite, and apatite affects successful milling of the ore and production of an acceptable or marketable concentrate. In addition, ore dressing is made more difficult when subsidiary minerals containing tin, molybdenum, and bismuth are present in sufficient quantities to warrant their recovery. To add further to milling problems, ores in tungsten deposits originating within the same general area or district may vary considerably in composition and characteristics (5, p. 17).

Most tungsten deposits are low grade, and the tenor of ore that can be profitably concentrated depends upon the market value of the concentrate per unit of tungsten and the amenability of the ore to concentration. Most mills strive to produce a concentrate containing at least 60 percent WO_3, but to achieve this with certain types of ore will occasionally result in lower recovery of tungsten. Generally, for scheelite ores treated in modern mills equipped with gravity and flotation facilities, 60- to 65-percent recovery is considered fair, 80-percent good, and over 90-percent exceptional. With wolframite ore 70-percent recovery is considered fair and 80- to 90-percent exceptional.

It usually is assumed that the higher the grade of mill feed, the higher the percentage of extraction that can be expected. This is not always true, for at times lower grade ores are more amenable to concentration than higher grade ores and produce more tungsten trioxide per ton of ore treated.

The capacity of a tungsten mill is rated by the number of tons of ore treated in 24 hours, the grade of ore, and the quantity of concentrate and tungsten trioxide produced (5, p. 17). The standard used in marketing tungsten is the unit (1 percent of a short ton or 20 pounds of WO_3).

Concentration of Tungsten Ores

Tungsten ores are beneficiated by crushing and grinding to liberation size of the tungsten minerals, followed by concentration of the tungsten minerals by gravity and/or flotation methods, leaching, roasting, and magnetic separation of the concentrate.

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Crushing and Grinding

All tungsten minerals are friable, and concentrating processes are hindered by the production of excessive fines. The first consideration in beneficiating or concentrating any tungsten ore is to recover the mineral without overgrinding, which causes excessive sliming. This is best accomplished by stage crushing and grinding.

Stamp mills are virtually obsolete. In a stamp battery tungsten minerals slime readily, because they are brittle. Their high specific gravity prevents their discharge from the battery unless they are finely divided (45, 46). Present practice favors primary and secondary crushing in jaw crushers. Where tungsten minerals can be liberated at coarse sizes, rolls can be used advantageously following secondary crushing, as they produce less slimes than grinding equipment. When finer grinding is required than is ordinarily achieved by rolls, a rod mill should be used as the first grinding step, as a rod mill ordinarily produces less slimes than a ball mill. For still finer grinding, a grate-type ball mill is used (46). Sliming in a ball mill can be lessened by using an excess amount of water to carry the ore through quickly, thus reducing the time of grinding contact in the mill. Grate-type ball mills have been used successfully at the Round Valley and Pine Creek tungsten mines near Bishop, Calif.

Classification

Because tungsten ores are friable and tend to slime, classification is a most important factor in achieving good recovery. To obtain a good recovery from tungsten ores, the fundamental practice that must be followed in modern milling is to size the ore as it passes through the plant and recover as much of the tungsten mineral as possible from each size material by using equipment best suited for each stage. This procedure is important because the sliming loss is reduced by minimizing the quantity of tungsten minerals present in the material subjected to regrinding. This method also reduces the amount of equipment necessary for treating slimes. The best practice in good classification and separation of slimes from sands (granular products) is to place in the mill circuit such screens, hydraulic classifiers, settlers, and cones as are consistent with economy, high recovery, and production of a marketable product.

Gravity Concentration

Jigging

Because of the high specific gravity (5.4 to 7.5) of tungsten minerals, gravity methods are commonly used to separate the tungsten minerals from the gangue material. Jigs are useful in concentrating tungsten ores in which the tungsten mineral occurs in a coarse form, permitting recovery of a coarse high-grade concentrate and thereby reducing slime losses resulting from additional grinding.

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Two types of jigs are commonly used. One is a pulsatory type that produces a finished coarse concentrate, a finished or semifinished hutch concentrate, and a coarse tailing. Such jigs include the Harz, Richards, and California-type Bendelari. The second type is the fine-mineral (hutch-type) jig, which produces a concentrate (hutch) and a tailing. Jigs of this type are the Denver and the Pan American.

Feed to the coarse-type jig usually consists of ore crushed from 1/4 to 3/4 inch in size, depending on the coarseness of the tungsten mineral in the ore. The size of the hutch product of the coarse-type jig varies with the size of the screen openings of the bed screen used in the jigging compartment and may range from 3/8 to 1/8 inch. Usually, the hutch product is a marketable concentrate; if not, it is upgraded in the fine jig, or by tabling.

The coarse jig tailing is usually treated in two ways before finer crushing. If the nature of the tailing material permits, the pulsatory stroke of the jig is shortened to permit a shorter and gentler lift in the jigging compartment. In this manner the coarse tailing is rejigged to produce a middling product and a final tailing for disposal. If this procedure is not permissible, all of the coarse tailing is recrushed. In either instance, the jig middling or the coarse tailing is recrushed in rolls to minus-10-mesh, and the unclassified, minus-10-mesh product is jigged in the fine-mineral jig. The fine-mineral jig recovers a high-grade concentrate in all sizes (except slimes) from the unclassified minus-10-mesh product. The final treatment consists of grinding and classifying the fine jig tailing in preparation for table concentration.

Tabling

Various types of shaking tables, such as the Wilfley, Deister, Overstrom, Isbell, Card, and James, which can be equipped with sand or slime decks, are used in concentrating tungsten ores.

If only coarse jigs are used in the circuit, the tungsten ore and/or jig tailing or middling is ground to minus-10-mesh in a ball mill or rod mill; the minus-10-mesh product is deslimed; and the sand product is classified into coarse, medium, and fine sands. The sized fractions are then concentrated separately on tables suitable for coarse, medium, and fine-sand treatment, producing a concentrate, a middling, and a tailing.

The middling from the coarse tables and sometimes the coarse tailing (if warranted) are reground and classified for further concentration on the medium and fine sand tables.

The same procedure is followed for the middling and tailing from the medium table. The amount of regrinding, classification, and retabling will depend, within economic limits, on the mesh grind necessary to unlock the tungsten minerals from the gangue material. The objective is to recover as much of the tungsten minerals as possible between each regrind and table treatment to avoid excessive losses of slime. Finally, the accumulated slimes are thickened and treated on a table provided with a slime deck or by flotation.

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Ample table capacity is required to reduce losses, as recovery is lowered decidedly if the tables are overloaded. Standardization on one make of table is preferred to accomplish flexibility in interchangeable decks and to minimize repair shutdowns.

Dewatering and Thickening

Mechanical thickeners are used for two purposes - first, to thicken the pulp to the desired density for further treatment; and second, to recover as much water as possible for reuse. Thickeners are essential in districts where water is scarce. Other means for thickening are cones, generally used for coarse products such as jig and table; and centriclones for classifier overflow products and slimes. Drag classifiers are used for dewatering coarse products, such as coarse concentrates, before drying.

Flotation Process

As late as 1931 flotation was largely limited to the separation of sulfide minerals from nonsulfide gangue minerals and for the selective separation of one sulfide mineral from another. Since then, several investigators (9, 10, 14, 15, 16, 20, 29, 30, 32, 36, 42) have successfully applied the flotation process to nonsulfide minerals, notably feldspar, phosphate rock, bauxite, fluorspar, rhodochrosite, manganese oxide, barite, siderite, chromite, kyanite, ferberite, and scheelite.

Coghill and Clemmer (9, p. 459) conducted extensive investigations on the floatability of various tungsten minerals and determined that scheelite may be floated readily from ores in which quartz gangue predominates. However, they found that the flotation of scheelite from contact metamorphic ores (tactite) containing a high percentage of calcite was more difficult, largely because both minerals (scheelite and calcite) contain CaO as the basic portion of the molecule. This similarity resulted in the flotation of calcite, as well as the scheelite, by the reagents that are effective for the scheelite, namely, oleic acid or sodium oleate. These reagents were of the bulk-flotation type and had no selectivity for the two minerals. Dean and Hersberger (14, p. 7) stated that calcite may be depressed by using certain concentrations of sulfuric acid. In practice, it is clear that it would not be economical to maintain a pulp at the required acid concentration with ores containing 25 percent or more of calcite.

Apart from the fact that the similarity between scheelite and the gangue mineral calcite in contact metamorphic ores causes difficulty in extracting the desired mineral selectively from the ore by flotation, the problem is complicated further by the necessity of making the separation on slimed material. Gaudin and Malezemoff (21, pp. 597-607) have shown that very fine, as well as very coarse, particles, were not readily floatable and that there was a definite reduction in floatability with reduction in particle size beyond a critical size of maximum floatability. These investigators believed that nonflotation of colloidal mineral particles was due to inability of the particles to come in contact with gas bubbles because of their fine size and state of dispersion. Flocculation, of course, overcomes the effect of excessive fineness

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of the particle, but it is necessary to flocculate selectively the valuable mineral particles without flocculating them with fine gangue particles. Coghill and Clemmer (9, p. 459) stated that the two phenomena - selective flocculating of the desired mineral and dispersion of the gangue - epitomize flotation; however, it is difficult to accomplish this on extremely small particles. Actually, the slime problem is one of utilizing the slimed material by the best possible method (either flotation or other means) after recovery of as much of the values as is economically possible by gravity methods. Usually, mills that utilize flotation as an adjunct to gravity separation grind the gravity middlings to flotation size and combine them with the slimes for flotation. In this manner, an additional recovery of 5 to 20 percent is achieved.

Apart from the limitations imposed by fineness of grind in treating slimes by flotation, as explained under Slimes and Their Characteristics, other factors (such as hardness of water, peculiarities of the ore, and local conditions) influence the recovery. Each area or district, although milling contact-metamorphic ores, has its own problem of beneficiation. Failure to accomplish economic recovery of the tungsten minerals from the slimes by flotation suggests recovery of the tungsten from the slimes by chemical means.

Flotation

Tungsten minerals are nonsulfide, and their concentration by flotation relies on the use of some derivative of the fatty acids. If contaminating sulfide minerals occur with tungsten, they are usually first removed by flotation using the usual reagent (such as xanthate, Reagent 301, or Reagent 404) or the aerofloats (such as Aerofloat 24, or Aerofloat 31, Reagent 238, or Reagent 242). The resulting tailing from the sulfide flotation is then treated with quebracho or palcotan, sodium silicate, and sodium carbonate, as gangue depressants, and with oleic acid or tall oil, as collectors for the tungsten minerals.

The tungsten content of most ores is low (0.4 to 1.0 percent WO_3), which means that high rejection of gangue material is essential. For this purpose, a lacy froth structure is preferred, which will support particles of tungsten minerals and at the same time permit the elimination of gangue material from the froth column. A suitable froth that accomplishes this is obtained by using sulfonated hydrocarbon reagents of the detergent and wetting-agent type, or certain rosin soaps.

Although the sulfonated reagents impart the required froth structure, they must be used in small quantities to avoid depressing the tungsten minerals. With some ores, even under the best conditions, the flotation concentrate is upgraded on shaking tables. Flotation concentrates that contain 6 to 20 percent WO_3 can be upgraded to 50 to 65 percent WO_3 by tabling. The table tailings that are too high in tungsten content to be discarded are returned to the circuit. Wetting agents aid in separating tungsten minerals from the gangue material during tabling operations.

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Promoters or Collectors

Promoters or collectors are reagents whose molecules will selectively attach to the surface of the mineral particles to be floated so as to cause the mineral to adhere to the froth bubbles and collect them in the froth as a concentrate. Chief among the promoters of tungsten minerals are fatty acids, soaps, and fatty acid emulsions. Oleic acid and sodium oleate are typical promoters. Oleic acid is widely used, both alone and in combination with Reagent 708, a fatty acid of vegetable origin. Emulsions of fatty acids stabilized with Aerosol OT or Aerosol 18 also may have possibilities as a promoter for certain tungsten minerals. Ten percent Aerosol OT has been used in the form of a dilute water solution as a spray on the froth just before the froth overflows the flotation machine. This treatment reduces the insoluble content of the concentrate (1, p. 22). Soaps, such as saponified linseed fatty acid or sodium oleate, are used with good results by some operators. Frequently, promoters that are effective on tungsten minerals may also promote other minerals not desirable in the tungsten concentrate. Reagent control is therefore of utmost importance.

Frothers

Most promoters or collectors used in the flotation of tungsten ores are frothers to a certain degree. The use of oleic acid, sodium oleate, or other soaps is often limited because of the excessive amount of froth they produce under conditions necessary for selectivity. Alcohols, cresylic acid, and Emulsol X-1 are commonly used with fatty acids. Other frothers and froth modifiers that may be used in the flotation of tungsten minerals are Aerosol 18, Aerosol 22, Aerosol OT, Reagent 801, and Reagent S-541.

When pure, frothers have no collecting properties; however, the presence of free fatty acids in any of them is important in flotation. The X-1 frother, in combination with oleic acid, is more effective for the flotation of scheelite than the X-2 series (Nos. 26 and 28) of frothers. Laboratory tests by Leaver and Royer (31), using X-1 in combination with oleic acid, achieved effective results in the flotation of scheelite, while X-2 members of the Emulsol froth series gave only fair results.

Modifying and Depressing Reagents

Soda ash, caustic soda, and mineral acids (such as hydrochloric and sulfuric) are used to control alkalinity or acidity of flotation pulps, as measured in hydrogen-ion concentration (pH). Soda ash is also used for softening water. Mineral acids sometimes are used to depress gangue minerals, and in an alkaline circuit sodium silicate is used with scheelite ores for this purpose. Sodium silicate, however, also has a depressing effect on wolframite, ferberite, and hübnerite. In a slightly acid circuit a metal salt, such as manganese sulfate, helps to prevent depression of ferberite by acids. Quebracho is a good depressant for quartz, calcite, and dolomite. A combination of sulfuric acid and sodium dichromate has been used for depressing calcite and other carbonates. A mixture of copper sulfate and sodium silicate is effective in depressing carbonates and fluorite. Aero brand cyanide or sodium cyanide can

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be used to depress sulfides during flotation of scheelite. Reagents 631 and 653 are helpful depressants and dispersants for micaceous gangue. Short-chain organic acids, such as formic, lactic, and mineral acids (including hydrochloric, nitric, sulfuric, or sulfurous acids), are sometimes used in depressing apatite, an objectionable phosphorus-bearing mineral. Most of the depressant reagents are added to the ball mill or conditioning tank.

Hydrogen-Ion Control

Various investigators seem to agree that the hydrogen-ion concentration in tungsten flotation is critical and that control of the pulp alkalinity within narrow limits is essential to provide the proper balance for successful flotation. However, some investigators speak of these narrow limits to be a pH between 8.0-8.5, others 9.0-9.5, and still others 9.5-10.5. The spread between 8 and 10.5 indicates that there is an optimum pH range for each particular ore within these limits, depending on the nature and characteristics of a particular ore and other influencing factors. It also implies that, once the optimum pH range is established for an ore, it must be maintained within narrow limits (possibly a spread of not more than one-half point of the optimum pH). The importance of close pH control on scheelite slimes is noted in the work by Leaver and Royer (31).

> Control of the pH of the pulp water within narrow limits is the main factor in successful recovery of scheelite by the flotation reagent mixture described. A minimum pH of 9.0 and a maximum one of 9.5 have been determined by the flotation results obtained over a period of several months. The correct pH is maintained by proper addition of soda ash to the pump lifting the pulp to the flotation machine. When the pH is below 9.0, the froth tends to carry siliceous material in addition to scheelite and some calcite. Higher pH than 9.5 has such a pronounced effect on froth subsidence that an excessive consumption of reagents for floating is necessary to maintain a freely flowing froth. These pH limits appear to provide the proper balance for successful flotation.

Roasting

Some tungsten mills use only a gravity circuit with no provisions for sulfide mineral flotation. The predominant sulfide mineral usually present in scheelite ores is pyrite. Because pyrite and scheelite have nearly the same specific gravity, they cannot be effectively separated from each other by gravity means. Pyrite can be separated from tungsten minerals in the concentrate by a magnetizing roast followed by magnetic separation.

A magnetizing roast is conducted at 600° C. or less. The object is to form an adherent magnetic film at the surface of the pyrite particles. This makes it possible to separate the pyrite particles from the concentrate when it is passed through magnetic fields as in magnetic separators.

Important factors in a magnetizing roast are size of particles, temperature, and time (48, pp. 176-177). Coarse particles require longer to heat to

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a required temperature than fine particles. The ignition temperature of pyrite must be maintained long enough for a film to form. Roasting time need not be prolonged beyond a few moments if the temperature is high enough.

The physical appearance of the product is a guide to roasting operations. A good magnetizing roast is indicated by the absence of pyrite particles and by a flat dead color, ranging from bronze to black. Black is characteristic of a deficiency of air during roasting or of a reducing atmosphere.

Furnaces of the rotary type, made of iron pipe and externally heated with oil or gas, are satisfactory for small operations. Rotary kilns or fluidized-bed reactors are satisfactory for large operations.

Magnetic Separation

Magnetic separators are useful adjuncts to gravity and flotation concentration of tungsten ores when tungsten ores or concentrates contain magnetic impurities, such as garnet, magnetite, ilmenite, and leucite, and nonmagnetic sulfide impurities, such as pyrite and pyrrhotite, that can be made magnetic by a magnetizing roast. These impurities, when present as discrete particles, can be separated from the nonmagnetic scheelite and gangue minerals. Magnetic separators are especially useful in separating wolframite from the tin mineral (cassiterite) and in separating magnetic wolframite, hübnerite, and ferberite minerals from nonmagnetic gangue minerals and scheelite. They are also useful in separating magnetic minerals from each other, based on the magnetic susceptibility of the mineral to be separated.

Table 3 gives the magnetic susceptibilities of a number of minerals, arranged in descending order of their attractive forces, using iron as a standard with an attractive force of 100. An easy separation of wolframite, with an attractive force of 5.68, from cassiterite, with an attractive force of only 0.13, is indicated. Similarly, garnet, with an attractive force of 6.68, and scheelite, with an attractive force of 0.15, indicate easy separation of these minerals. However, the attractive force of 0.23 for apatite and 0.15 for scheelite indicate a close range of attractive forces necessitating extremely close control of the magnetic forces to effect separation of the two minerals that, in all probability would not be practical.

Two types of dry magnetic separators (induction roll and belt-type) are in common use in beneficiating tungsten ores or concentrates. Induction-roll-type separators include the Dings, Stearns, and Exolon. The induction-roll type gives a more extensive field, which promotes a cleaner tailing reject; the belt- or Wetherill-type separator gives higher selectivity. The selection of the induction-roll type, belt type, or a combination of both depends largely on the type and number of impurities to be removed from the ore or concentrate. For simple removal of garnet or pyrite from scheelite, the induction-roll type is satisfactory. For separating various magnetic minerals from each other, selectivity is required, and the Wetherill-type separator (equipped with the required number of crossbelts and magnets with separately controlled current) would be more desirable. For example, in treating beach sands containing magnetite, ilmenite, chromite, and zircon, magnetite is removed by the first

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magnet at low intensity and ilmenite by the second magnet at higher intensity than the first magnet, leaving chromite to be lifted from the zircon by the third and most powerful magnet. Similarly, in treating tungsten minerals, fragments of crusher steel, magnetite, garnet, etc., must be removed by comparatively weak magnets before the strongest magnet is applied to separate the more valuable but weakly magnetic minerals. Middlings or particles containing a mixture of minerals and gangue must sometimes be removed for further grinding and other preliminary treatment.

TABLE 3. - Magnetic susceptibility of minerals[1/]

Material	Attractive force	Material	Attractive force
Iron.....	100.00	Argentite.....	0.27
Magnetite.....	49.19	Orpiment.....	.24
Franklinite.....	33.49	Pyrite.....	.23
Leucite.....	17.50	Sphalerite.....	.23
Silicon.....	17.42	Molybdenite.....	.23
Pyrrhotite.....	15.43	Apatite.....	.23
Ilmenite.....	11.67	Bornite.....	.22
Biotite.....	8.90	Willemite.....	.21
Garnet.....	6.68	Tetrahedrite.....	.21
Wolframite.....	5.68	Scheelite.....	.15
Hematite.....	4.64	Talc.....	.15
Limonite.....	3.21	Arsenopyrite.....	.15
Chromium.....	3.12	Magnesite.....	.15
Pyrolusite.....	2.61	Chalcopyrite.....	.14
Rhodochrosite.....	1.93	Cassiterite.....	.13
Siderite.....	1.82	Gypsum.....	.12
Corundum.....	1.40	Fluorite.....	.11
Manganite.....	1.36	Zincite.....	.10
Rutile.....	.93	Chalcocite.....	.09
Platinum.....	.76	Cuprite.....	.08
Dolomite.....	.57	Orthoclase.....	.05
Calamine.....	.51	Cryolite.....	.05
Quartz.....	.40	Galena.....	.04
Cerussite.....	.30	Calcite.....	.03
Cerargyrite.....	.28		

1/ Magnetic Engineering & Manufacturing Co. (Memco), Catalog No. 552.

The essential requirements for clean magnetic separation (13) of a product from a material are that the material treated must be thoroughly dry (wet, fine materials stick to belts or rolls) and mechanically free and that the desired product must have magnetic properties that differ from those of other constituents from which it is to be removed. For differential separation of two or more magnetic substances, sizing of the material is necessary, as is shown by considering the forces acting on large and small particles in a magnetic field.

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The limit of fineness that can be magnetically treated depends on the nature and characteristics of the ores; some respond to treatment as fine as 280-mesh, whereas others give trouble at 100-mesh.

Air Tabling

Pneumatic methods of concentrating tungsten ores have been used to a limited extent. From 1898 to 1901 the Trumbull mine near Long Hill, Conn., produced the first tungsten concentrates in the United States by using this method. The tungsten minerals, principally scheelite, occurred at a contact of limestone and gneiss. The associated minerals were chiefly garnet, quartz, hornblende, epidote, and pyrite. The ore, which assayed about 5 percent scheelite, was stage-crushed with rolls and the product classified by screening into 40-, 60-, and 90-mesh fractions. These products were fed to six Hooper pneumatic machines that delivered a clean product, leaving little scheelite in the tailings. Hobbs (28, pp. 13-22) states that each machine could handle 10 tons of dry material per day, yielding a final product containing 65 percent WO_3.

In 1924 a small pneumatic concentrating test plant was erected on the Summerfield property near Mina, Nev. (36, pp. 287-289). The ore contained scheelite disseminated through beds of garnetized limestone. At this plant, the ore was crushed and ground in a ball mill to minus-10-mesh. The ball-mill discharge was fed to a Stebbins dry table that had a capacity of 1 ton of ore per hour. The table yielded three products - a scheelite concentrate containing 15 percent WO_3, a middling product that was returned to the ball mill for regrind, and a waste tailing. The scheelite concentrate was cleaned on a smaller Stebbins table that produced a concentrate containing 60 percent WO_3.

Approximately 500 tons of ore was treated in the test plant. The grade of the ore ranged up to 4.5 percent WO_3, but the average was only 1.1 percent WO_3, which at the time did not permit profitable operations on a small scale; the project was therefore suspended. Table 4 is a screen analysis of the mill feed and tailing during one of the test runs in which the mill feed averaged 1.98 percent WO_3.

TABLE 4. - Screen analysis of mill feed and tailing, Summerfield property

	Mill feed		Mill tailing	
Size, mesh	Weight of total, percent	Analysis WO_3, percent	Weight of total, percent	Analysis WO_3, percent
+20.....	36.3	0.83	41.7	Nil.
+40.....	28.5	1.10	27.0	Trace.
+60.....	13.4	3.17	13.0	0.90
-60.....	21.5	4.33	18.3	1.37
Average.....		1.98		.36
Mill recovery..	81.89 percent			

The results shown in the table indicate that air tables are effective in concentrating high-grade scheelite ores that contain grains of coarse

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scheelite in the plus-60-mesh range but are not suitable for fine scheelite particles. Since wet tables and flotation are much more effective for concentrating fine material, air tables have not been considered a concentrating medium in modern tungsten mills. However, air tables are useful in arid regions where water is scarce, and the grade and characteristics of the ore permit economic recovery of the scheelite.

Heavy-Medium Separation

The heavy-medium process, often referred to as the sink-and-float process for mineral concentration, has advanced appreciably since its appearance as a commercial process in 1936. Since then, vessel designs other than the cone have been introduced and include classifier and drum-type separators. The range size of the feed amenable to treatment has expanded to include materials as large as 10 inches and as small as 28-mesh. Today, more than 20 types of ore are being treated by this method, among them ores of iron, lead, zinc, lead-zinc, fluorspar, tin, magnesite, dolomite, chromite, andalusite, and manganese and coal. In addition, some gravel plants utilize the process for separating light and heavy aggregate.

With regard to application of the heavy-medium process to tungsten ores, a heavy-medium pilot plant at Barking, England (37), has experimented with galena as a suspension medium for beneficiating tungsten ores. The Pan American Tungsten Co. has installed a heavy-medium mill at the Isle of Pines, Cuba, for treating tungsten ores, using ferrosilicon as the medium. More recently, the Hualpai Mining Co. at Yucca, Ariz., was licensed to use the heavy-medium process for treating tungsten ore from the Boriana mine.

The range of specific gravities among tungsten minerals, associated minerals, and gangue material in most tungsten ores is quite large. For example, the specific gravities of the economic tungsten minerals are: Scheelite, 5.9-6.1; wolframite, 7.2-7.5; ferberite, 7.5; and hübnerite 7.2-7.5. Those of the associated gangue minerals are: Magnetite, 5.2; garnet, 3.2-4.3; pyrite, 5.0; psilomelane, 3.7-4.7; siderite, 3.9; sphalerite, 3.9-4.1; and apatite 3.2. Those of the gangue materials are: Calcite, 2.7; dolomite, 2.9; epidote, 3.2-3.5; and quartz, 2.7.

According to Pearson (37, p. 7), the flexibility of the heavy-medium process is such that minerals having close specific gravities can be separated from each other, provided that the minerals can be freed from their gangue and associated minerals at a size range within the limits of the heavy-medium process. Since the scheelite that occurs in most contact-metamorphic deposits is fine grained, the process does not lend itself for concentrating the scheelite. However, application of the process for tungsten ores that are amenable to it should not be overlooked in a mill circuit. If its application did nothing more than effect preconcentration of the tungsten minerals, eliminating 60 to 70 percent of the barren material between the crushing and grinding step, it would effect a substantial economy in the number and size of other equipment necessary to obtain the same units of tungsten.

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BENEFICIATION RESEARCH

Recovery of Tungsten Minerals by Flotation and Purification by Acid Leaching

For many years, beneficiation of tungsten ores consisted largely of hand-picking and gravity-concentration processes. Ores that were amenable to gravity concentration at relatively coarse sizes usually yielded a high-grade concentrate but incurred high losses in the fines. Complex ores requiring fine grinding for liberating the tungsten minerals were not treated successfully by gravity methods. The need for better methods or a combination of methods to achieve economic recovery of tungsten minerals from various types of tungsten ores was apparent.

Fundamental studies in application of flotation for treating various tungsten ores were initiated by the Bureau of Mines in 1930. Ores containing scheelite, ferberite, wolframite, hübnerite, and complex ores (containing two or more minerals) were used in the studies. Oleic acid and sodium oleate were used for flotation of tungsten minerals; sodium silicate was used to depress siliceous gangue material, formic, lactic, and mineral acids were used for depressing apatite. Copper sulfate was used to condition the pulp following a sulfide flotation. Flotation concentrates, high in apatite and calcite, were leached with hydrochloric acid. The results of the tests discussed herein are based on the work of Clemmer and O'Meara (8, p. 15) in the application and adaption of flotation to tungsten ores.

In this investigation, three types of tungsten ores were studied: (1) A scheelite ore from Nevada, predominantly quartz with a small amount of apatite; (2) a scheelite ore from Montana, containing sulfides, gold, arsenopyrite, mica, apatite, and quartz; (3) a scheelite ore, containing calcite and other associated carbonate minerals and calcareous gangue from a contact-metamorphic deposit.

For the Nevada scheelite ore, oleic acid and sodium oleate were used to float the scheelite, and a small amount of sodium silicate was used to depress the quartz. A satisfactory separation was achieved. The concentrate was high grade and represented a good recovery of the tungsten. The phosphorus (apatite) content, however, exceeded allowable limits. A final cleaning with formic acid depressed the apatite; 96.7 percent was rejected. The best results were achieved when just enough acid was added to the cleaner to make the pulp neutral or slightly acid. Table 5 shows the results of this work.

For the scheelite ore from Montana the arsenopyrite was floated in an acid circuit with the customary sulfide flotation reagents. Subsequently, the scheelite was floated with oleic acid and sodium oleate. A small amount of sodium silicate was added to retard the quartz. The rougher froth was cleaned with sodium silicate to depress quartz and mica. A final cleaning with formic and lactic acids depressed the remaining mica and apatite. Lactic acid was especially effective in depressing mica in the flotation of both sulfide and scheelite. It also depressed apatite but to a lesser degree than the inorganic

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or short-chain organic acids. In the absence of the phosphorus-bearing mineral (apatite) an alkaline circuit was preferable; but when apatite was present, an acid circuit was essential. Table 6 shows the results of this test.

For the calcareous, contact-metamorphic ore, the objective was to separate scheelite from calcite and other calcareous gangue minerals. Several flotation tests were made, and the results of a typical test are given in table 7. Scheelite was concentrated from ores containing calcareous gangue to give a high-grade product, low in impurities; the roughing and cleaning steps rejected 98.6 percent of the phosphorus.

TABLE 5. - Flotation of a Nevada scheelite quartz ore, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	S	WO_3	P
Concentrate	3.6	77.06	0.036	0.03	87.6	3.3
Middling	3.9	9.54	.372	-	11.5	35.8
Tailing	92.6	.03	.026	-	.9	60.9
Composite	100.0	3.16	.040	-	100.0	100.0

Reagent	Pound per ton of crude ore			
	Rougher	Cleaner 1	Cleaner 2	Cleaner 3
Sodium silicate	0.40	0.15	0.15	-
Oleic acid	.32	-	-	-
Sodium oleate	.2	-	-	-
Formic acid	-	-	-	0.60

TABLE 6. - Flotation of a Montana scheelite ore containing sulfides, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	As	WO_3	P
Sulfide concentrate	5.2	1.44	-	-	2.0	-
Scheelite concentrate	4.2	77.91	0.032	0.056	89.8	0.21
Middling	2.5	9.94	-	-	6.8	-
Tailing	88.1	.05	-	-	1.4	-
Composite	100.0	3.64	-	-	100.0	100.0
Feed	-	3.62	.636	1.18	-	-

Reagent	Pound per ton of crude ore					
	Sulfide			Scheelite Cleaner		
	Conditioner	Rougher	Rougher	1	2	2
Sulfuric acid	1.00	-	-	-	-	-
Lactic acid	.12	0.08	-	-	-	-
K-amyl xanthate	-	.35	-	-	-	-
Hardwood creosote	-	.20	-	-	-	-
Oleic acid	-	-	0.32	-	-	-
Sodium oleate	-	-	.10	-	-	-
Sodium silicate	-	-	.30	0.15	0.10	-
Tannic acid	-	-	-	-	-	0.40

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TABLE 7. - Flotation of scheelite from contact metamorphic ores, 65-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	As	WO_3	P
Sulfide concentrate.....	16.0	3.74	0.082	-	7.0	10.7
Scheelite concentrate.....	7.8	74.10	.022	0.03	68.2	1.4
Middling.....	11.8	14.64	-	.249	20.3	24.1
Tailing.....	64.4	.60	.121	-	4.5	63.8
Composite.....	100.0	8.49	.122	-	100.0	100.0
Feed.....	-	8.54	.137	5.27	-	-

Reagent	Pound per ton of crude ore				
	Sulfide rougher	Scheelite			
		Conditioner	Rougher	Cleaner 1	Cleaner 2
K amyl xanthate.....	0.40	-	-	-	-
Aerofloat.....	.32	-	-	-	-
Copper sulfate.....	-	0.8	-	-	-
Sodium silicate.....	-	-	1.0	0.25	0.20
Oleic acid.....	-	-	.96	-	-
Sodium oleate.....	-	-	.20	-	-

The tungsten ores used in these tests varied considerably in grade and, hence, the amount of reagents used varied with the ore tested. In every instance, however, good concentrates were attained with satisfactory recovery of tungsten.

The procedure sequence used was to float the sulfides, condition the tailing pulp, float the nonsulfide minerals, and clean the rougher froth. The sulfide minerals were floated using the customary sulfide reagents. Since they must be thoroughly removed to permit production of a tungsten concentrate low in sulfur in the subsequent tungsten flotation step, relatively large quantities of sulfide float reagents were required. A 10- or 15-minute flotation period usually was ample to remove the sulfide minerals from either acid or alkaline pulps. For nonsulfide or scheelite flotation the tailing pulp from the sulfide flotation step was conditioned several minutes with copper sulfate, a heavy metal salt. Other metal salts, such as copper nitrate or copper ammonium hydroxides, may be used. The metal salt is added to the conditioner, or it may be added to the sulfide flotation circuit, provided the salt has no retarding effect on the sulfides. A soluble silicate was then added; and, after a period of conditioning the pulp, oleic acid and sodium oleate were added to float the scheelite. The calcite and apatite were retarded by the metal-silicate combination. The scheelite rougher froth was of good grade. The quantities of reagents required depended on the nature and grade of the ores and were determined experimentally.

The duration of the silicate-conditioning period was most important because a short period resulted in only a fair recovery of tungsten and high-grade concentrates that are low in phosphorus, while a longer period (5 to 10 minutes) gave increased recovery of low-grade concentrates but failed to reject the apatite thoroughly. Within certain limits a decreased amount of soap permitted production of high-grade concentrates without a serious sacrifice in recovery.

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Cleaning the rougher froth one or more times with sodium silicate or an acid reagent depressed the residual calcite or apatite floated in the rougher operation. In the reported test (table 7) two cleanings were applied, using sodium silicate. Rougher froths from tests in which a proper quantity of metal salt-silicate reagent was used were readily cleaned with sodium silicate alone. Failing to use the proper amount of reagents, final cleaning in a neutral or slightly acid pulp is necessary to depress the remaining calcite and apatite.

In the reported test no alkali other than sodium silicate was used in the scheelite rougher. In other tests improved recovery was obtained when caustic soda or soda ash was used with the silicate without sacrificing the grade of concentrate. In one test 86 percent of the scheelite was recovered in a concentrate assaying 70 percent tungstic acid (WO_3). In another test a comparison was made in using a sodium silicate-dichromate combination as against a metal salt-silicate combination. The test shows that, although the recovery was satisfactory, the concentrate was lower in grade than that obtained by the metal salt-silicate combination.

In addition to depressing the apatite while floating scheelite, attempts were made with several scheelite ores to float the apatite before the scheelite was floated. The latter were only moderately successful. A small amount of soap yielded an apatite concentrate, but the percentage of total phosphorus removed was small. Any advantage gained was offset by the loss of tungsten in the apatite concentrate. Increasing the amount of soap to float the apatite was accompanied by additional losses of tungsten in the apatite concentrate.

An alternative method to removing the apatite rather than to depressing it in flotation was to leach the flotation concentrate with dilute hydrochloric acid. Several leaching tests with acid were made on flotation concentrates containing apatite. In each instance the apatite was dissolved, and low-phosphorus residues were produced; no appreciable quantity of tungsten was dissolved. The amount of acid required for leaching depended on the grade of the concentrate and on the nature and quantity of associated gangue minerals. A siliceous ore required less acid than one containing calcite and carbonates.

A siliceous concentrate assaying 0.295 percent phosphorus was leached with a commercial hydrochloric acid solution that contained 25 pounds of acid per ton of concentrate. The resulting residue contained 0.007 percent phosphorus, indicating 97.6-percent extraction of the phosphorus. The loss of tungsten was negligible.

For a concentrate produced from a contact-metamorphic ore containing calcite and apatite appreciably more acid was required to leach the calcite and apatite. An excess of acid was necessary to insure complete solution of the apatite. A concentrate containing 0.266 percent phosphorus was leached with a slight excess of acid, leaving a residue containing 0.09 percent phosphorus.

In another investigation ferberite, wolframite, hübnerite, and complex ores were studied. Most domestic ferberite ores have a predominantly siliceous gange and only minor sulfide minerals, siderite, and apatite. Coarsely

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crystalline ores are concentrated by gravity methods; but ferberite is very brittle, and crushing produces rich fines that are difficult to recover by gravity means.

Ferberite was floated in an alkaline pulp with oleic acid and sodium oleate. Soda ash or caustic soda was used with silicate to depress the siliceous gangue. High-grade concentrates were obtained, with a satisfactory recovery. Cleaning with acid depressed the apatite and gave a low phosphorus concentrate. In the flotation of a low-grade jig concentrate produced from a Colorado ferberite ore, acid was used in the rougher and cleaners; acidified dichromate was also very effective. Table 8 gives the results of this test.

TABLE 8. - Flotation of a low-grade jig concentrate, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	S	WO_3	P
Concentrate.....	38.3	62.38	0.012	0.031	76.7	20.9
Middling.....	28.8	23.96	.032	-	22.1	50.3
Tailing.....	32.9	1.10	.016	-	1.2	28.8
Composite.....	100.0	31.15	.018	-	100.0	100.0
Feed.....		31.20	.022	.019		

Reagent	Pound per ton of crude ore			
	Rougher	Cleaners		
		1	2	3
Oleic acid.....	0.88	-	-	-
Sodium oleate.....	.35	-	-	-
Acid dichromate.....	.05	0.15	0.25	0.30

In certain ores in which the ferberite was sensitive to depression by acid, introduction of a small amount of metal salt, such as manganese sulfate, rendered the mineral less sensitive without reducing the gangue-depressing property of the acid. The metal-salt combination also depressed apatite. Most ferberite ores were relatively free of carbonates. If small amounts were present, they were depressed with the apatite in a slightly acid circuit. Ferberite flotation in an acid, rather than an alkaline, circuit gave the best results, although ferberite was not as floatable as scheelite.

Wolframite ores were a little more floatable than ferberite. The reagents used for ferberite acted similarly on wolframite ores. A slightly acid circuit was preferable for flotation of wolframite. Alkaline circuits were tried but failed to reject the phosphorus as thoroughly as the acid circuit. Table 9 gives the results of a typical test on a high-grade Washington wolframite.

Hübnerite was more floatable than either wolframite or ferberite. Reagents that were suitable for wolframite or ferberite were also satisfactory for hübnerite ores. Table 10 gives the results of a typical test of an Arizona hübnerite ore containing small quantities of phosphorus and sulfide minerals. The sulfides were mainly pyrite and chalcopyrite. They were floated first with xanthate. Butyl alcohol was preferred as frother, although some silicate was used in the sulfide rougher to prevent the tungsten from floating with the sulfides. Hübnerite was then floated with oleic acid and sodium

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oleate. The rougher froth was cleaned twice, using acidified dichromate in the second cleaner to depress the remaining quartz and apatite.

TABLE 9. - Flotation of a high-grade wolframite ore, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	S	WO_3	P
Concentrate.....	34.5	70.92	0.013	0.01	89.4	29.6
Middling.....	21.7	12.54	.027	-	10.0	38.6
Tailing.....	43.8	.39	.11	-	.6	31.8
Composite.....	100.0	27.36	.015	-	100.0	100.0
Feed.....	-	28.69	.015	.002	-	-

Reagent	Pound per ton of crude ore		
	Rougher	Cleaner 1	Cleaner 2
Oleic acid.....	0.95	-	-
Sodium oleate.....	.40	-	-
Acid dichromate.....	.05	0.10	0.15

TABLE 10. - Flotation of Arizona hübnerite ore, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total	
		WO_3	P	S	WO_3	P
Sulfide concentrate.	4.2	1.31	0.016	-	1.4	8.7
Hübnerite.....	5.3	71.56	.012	0.043	90.2	8.2
Middling.....	1.2	10.24	.015	-	2.9	2.3
Tailing.....	89.3	.26	.007	-	5.5	80.8
Composite.....	100.0	4.20	.008	-	100.0	100.0
Feed.....	-	4.25	.005	-	-	-

Reagent	Pound per ton of crude ore			
	Sulfide	Hübnerite rougher	Cleaner 1	Cleaner 2
X-amyl xanthate.....	0.25	-	-	-
Sodium silicate.....	.65	-	-	-
Butyl alcohol.....	.14	-	-	-
Oleic acid.....	-	0.40	-	-
Sodium oleate.....	-	.10	-	-
Acid dichromate.....	-	-	-	0.25

With regard to selective separation of tungsten minerals in a complex ore, the order of increasing floatability of tungsten minerals is: Ferberite, wolframite, hübnerite, and scheelite. One would expect that scheelite could be separated from wolframite. This separation has been made, although such a separation would not be advantageous on tungsten ores in which scheelite and wolframite were the only readily floatable minerals. If the ore contained other floatable minerals, such as fluorspar, a separation would be necessary to produce a marketable product. The fluorspar and scheelite could be concentrated into one product and the wolframite into another. Re-treatment of the fluorspar-scheelite product would yield a fluorspar concentrate and a scheelite concentrate.

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In an ore from the Springvale mining district, Washington, the tungsten minerals present were wolframite and scheelite, associated with sulfide minerals. The principal gangue minerals were quartz and mica. Other associated minerals were fluorspar, calcite, ankerite, apatite, and tourmaline. The complex nature of the ore and the low tungsten content (1.48 percent WO_3) made it difficult to achieve selective separation of the tungsten minerals, but several tests showed that the ore could be concentrated by flotation. Results from a typical test are given in table 11, and the flowsheet is shown as figure 5.

TABLE 11. - Flotation of complex tungsten ores, 150-mesh

Product	Weight-percent	Analysis, percent			Percent of total
		WO_3	P	S	WO_3
Sulfide concentrate	1.9	2.09	-	-	2.7
Fluorspar concentrate	2.4	1.53	-	-	2.5
Scheelite concentrate	.5	59.85	0.051	0.020	20.2
Wolframite concentrate	.9	68.32	.020	.050	41.5
Middling	5.6	8.44	-	-	32.0
Tailing	88.7	.02	-	-	1.1
Composite	100.0	1.48	-	-	100.0
Feed	-	1.48	.009	-	-

	Pound per ton of crude ore							
		Scheelite			Wolframite			
	Sulfide	Cleaner			Cleaner			
Reagent	rougher	Rougher	1	2	Rougher	1	2	3
K amyl xanthate	0.30	-	-	-	-	-	-	-
G N S No. 17	.10	-	-	-	-	-	-	-
Sodium silicate	.20	-	0.35	-	-	-	-	-
Oleic acid	-	-	-	-	0.12	-	-	-
Sodium oleate	-	0.1	-	-	.03	-	-	-
Formic acid	-	-	-	0.25	-	-	-	-
Hydrochloric acid	-	-	-	.10	-	-	0.10	0.15
Manganese sulfate	-	-	-	-	-	0.2	-	-
Sulfuric acid	-	-	-	-	-	.2	-	-
Lactic acid	-	-	-	-	-	-	.10	-

MILLING PRACTICES AT PRINCIPAL MINES

The milling practices described herein and prepared from a survey made in 1955, represent methods used during that time and up to December 1956, when Government purchasing of tungsten ores and concentrate ceased and most operations were suspended.

The reader will note that many of the flowsheets or figures accompanying the description of the milling practices contain numbers. To avoid confusion and to conform with the text, these numbers are used to indicate the position of equipment and not the number of pieces or size of equipment.

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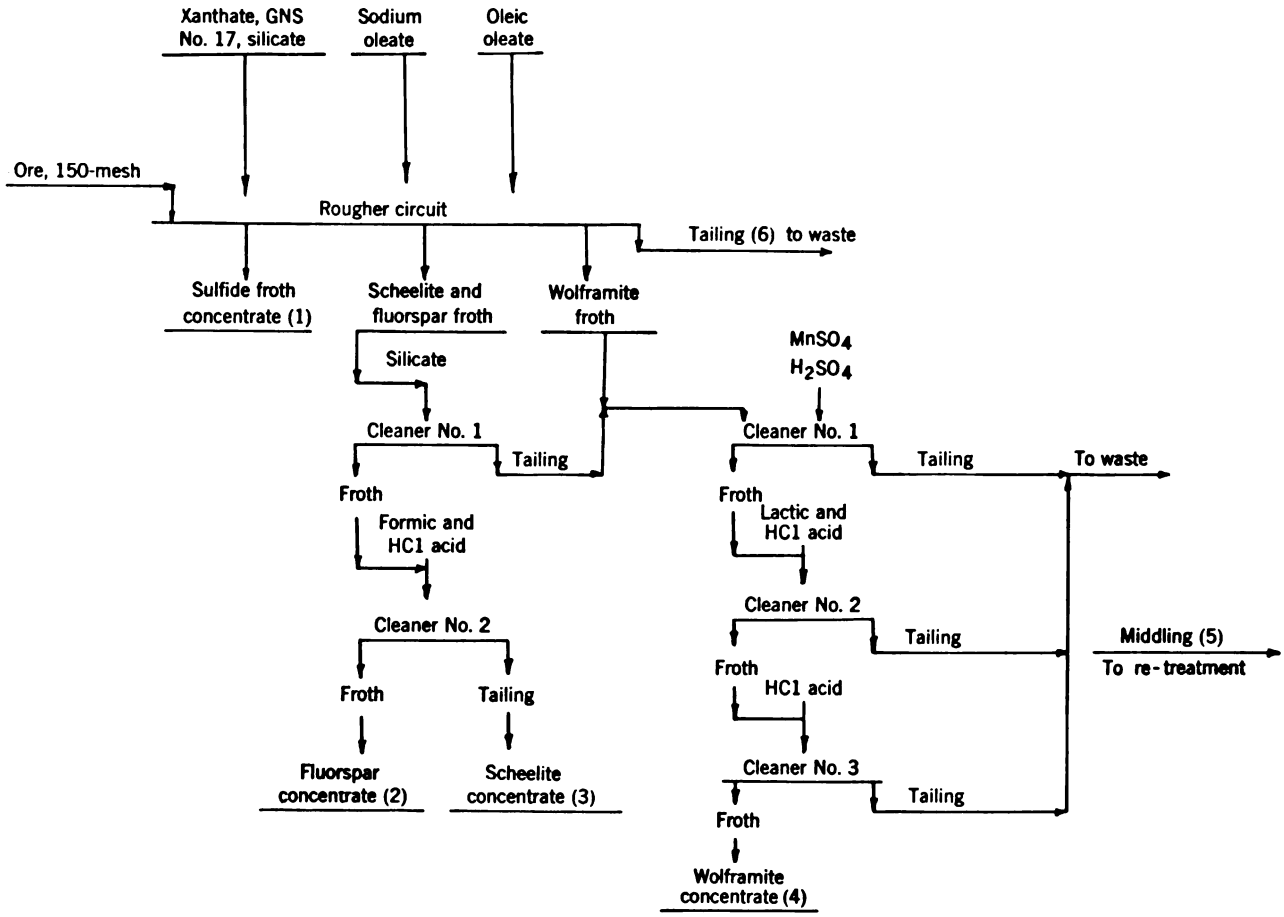

FIGURE 5. - Flowsheet for Treatment of Complex Tungsten Ores.

Getchell Mine, Inc.

The mines and mill of the Getchell Mine, Inc., are 23 miles north of Golconda, Humboldt County, Nev. Approximately 60 percent of the ore treated is truck-hauled from six company deposits; the remainder is custom ore, truck-hauled to the mill from numerous mines within a radius of 200 miles. The ores in the area usually occur as tungsten-bearing tactites in contact-metamorphic deposits. Among the identifiable minerals in the ore are: Scheelite, powellite, apatite, garnet, epidote, calcite, pyrite, molybdenite, quartz, and graphite. The tungsten mineral is concentrated by flotation, followed by a two-stage acid leaching to eliminate calcite and apatite. Mill capacity and tonnage treated during 1955-56 were at the rate of 1,200 tons of ore per 24 hours, producing a final concentrate containing more than 65 percent tungsten trioxide at an optimum recovery. Figure 6 is the flowsheet of the milling operation.

Crushing and Grinding

The ore produced at the Getchell deposits is hauled in 15-ton tandem, dual, rear-dump, Kenworth trucks, powered with Cummins diesel engines. At the

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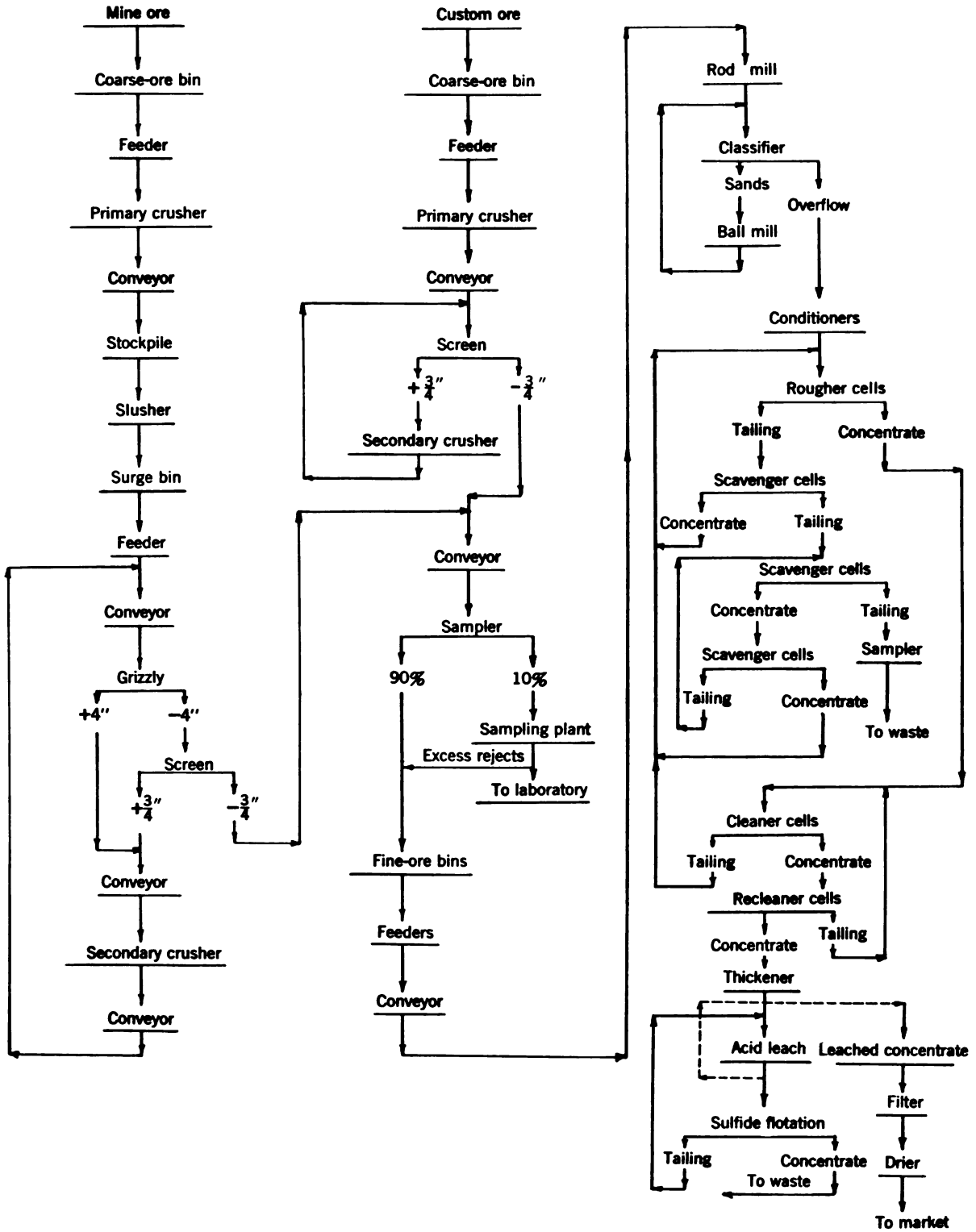

FIGURE 6. - Flowsheet, Getchell Mill, Getchell Mine, Inc.

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mill, a 42- by 48-inch Traylor jaw crusher and a 4-foot Traylor TY secondary crusher are used for company ore. Custom ore is crushed in a separate crushing plant, with an 18- by 36-inch jaw crusher and a 3-foot TY secondary crusher. After crushing to minus-3/4-inch, the ore from both plants is conveyed to the sampler, then to two 1,000-ton fine-ore bins.

In the grinding circuit special consideration is given to minimize overgrinding. Analysis of tailing (table 12) shows that 13.8 percent of the tungsten carried in the waste is contained in the minus-100, plus-200-mesh fraction, and 80.8 percent in the minus-200-mesh fraction. Because relatively small amounts of the tungsten in these particle sizes can be economically recovered in normal flotation operation, it is important that overgrinding be kept to a minimum. Optimum grinding is achieved by a combination of an 8- by 12-foot rod mill in open circuit, followed by an 8- by 6-foot ball mill in closed circuit with a classifier. To minimize overgrinding further, the classifier overflow is maintained between 53 and 56 percent solids. The grind is approximately 10 percent plus-48-mesh. The overflow is pumped to four conditioning tanks that have 30-minute contact time before flotation. Table 12 gives a typical screen analysis of the mill feed and tailing.

TABLE 12. - Typical screen analysis, Getchell Mine, Inc.

	Mill feed			Mill tailing	
Mesh	Weight percent	WO_3, percent	WO_3 distribution, percent	Weight-percent	WO_3 distribution, percent
+48.....	10.4	0.06	1.62	6.3	0.33
65.....	11.5	.08	2.40	10.4	3.27
100.....	13.0	.13	5.18	11.1	1.74
150.....	10.7	.39	10.85	-	-
200.....	8.7	.55	12.45	18.9	13.84
325.....	11.2	.56	16.33	10.7	.56
-325.....	34.5	.57	51.17	42.6	80.26
Composite....	100.0	.384	100.00	100.0	100.00

Flotation

The conventional flotation of sulfide minerals before fatty acid flotation is omitted in the mill circuit. Sulfide minerals are removed, however, by flotation of the acid-treated concentrate, as described under Details of Leaching Procedure at Getchell. In the fatty acid flotation of tungsten minerals, the pulp from the conditioners has a pH between 10.2 and 10.4 goes to a rougher circuit that consists of four 66-inch Booth cells and is floated at a pulp density of 34 percent solids. The rougher froth is cleaned in a bank of No. 18 Denver cleaners. The tailing goes to a scavenger circuit, consisting of four 66-inch Fagergren cells, five 54-inch Booth cells, and three 54-inch Fagergren cells. The concentrate from the four 66-inch Fagergren cells is returned to the head of the rougher circuit and the tailing to the bank of five 54-inch Booth cells. Tailing from the Booth cells is automatically sampled and discarded to the tailing pond. The concentrate goes to the three 54-inch Booth cells, and the concentrate is brought back to the head of the rougher circuit.

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In the cleaner circuit the rougher concentrate is cleaned twice in a bank of four No. 18 Denver cleaner cells. The tailing from the cleaner circuit is returned to the head of the rougher circuit, and the froth from the first three cells is recleaned in the fourth cell. Cleaning is accomplished at a pulp density of 15 percent solids. The final concentrate is pumped to a 44-foot thickener to await acid treatment, and the tailing is recirculated in the cleaner circuit. The final cleaned concentrate averages 30 to 40 percent tungsten trioxide.

Flotation is conducted in water at 74° to 80° F. The temperature of the mill feedwater increased approximately 20° F. with steam supplied at 15-pound pressure from a 150-hp. Kewanee boiler through a coil heat exchanger. The fuel used is Bunker C oil that costs nearly 5 cents per ton of ore milled. The heat exchanger delivers 400 gallons of water per minute. The warm-water flotation circuit increases recovery of tungsten approximately 4 percent.

To insure maximum rejection of undesirable minerals in the concentrate, the condition of the froth is closely controlled. Even a minor excess of oleic acid produces a thick, sticky froth that markedly reduces concentrate grade. A lacy froth is sought continuously. Because of the changing pulp characteristics resulting from the variety of custom ores treated, the quantity of reagents required to maintain optimum concentration efficiencies changes several hundred percent during the course of a shift. Much of the operator's time is spent observing the froth and devising reagent combinations that will effectively meet the changing conditions. Pilot tables, together with visual observation of froth characteristics, enables the operator to maintain optimum mill operations. Other influencing factors are soluble salts in the ore or mill makeup water that react with oleic acid to form metal oleates. These cause high tailing losses or lower grade concentrate; however, this condition is corrected by increasing the amount of soda ash fed to the grinding circuit. Table 13 gives a typical analysis of the final concentrate. Table 14 provides the flotation-reagent consumption and places of addition.

TABLE 13. - Typical analysis of final acid-leached concentrate, Getchell Mine, Inc.

Concentrate	Analysis, percent	Concentrate	Analysis, percent
Tungsten trioxide.	65.38	Molybdenum.....	2.93
Tin.....	Trace	Phosphorus.....	.02
Copper.....	.02	Sulfur.....	.18
Arsenic.....	None	Manganese.....	.11
Antimony.....	Do.	Lead.....	.02
Bismuth.....	Trace	Zinc.....	.02

TABLE 14. - Flotation-reagent consumption and place of addition, Getchell Mine, Inc.

Reagent	Pounds per ton	Place of addition
Soda ash.....	5.0-18.0	Rod mill
Palcotan.....	.16- .25	Sulfide circuit
Sodium silicate.....	2.5-6.0	Rod mill
Oleic acid.....	.2- .6	Classifier overflow
D-40 detergent.....	.15-.3	Head of flotation
Frother 77.....	.1- .2	Sulfide circuit

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Two-Stage Leaching of Scheelite Concentrate

As in other plants where hydrochloric acid is used to digest the gangue minerals, and particularly where the phosphorus content is high, the Getchell mill employs a two-cycle leaching process. The first leach removes calcite; the second removes phosphorus. Close control is required to prevent losses, as scheelite is soluble in the acid.

During the first leach, violent effervescence occurs. The heat of the reaction is sufficient to raise the temperature of the pulp; but, in spite of the heat generated, very little scheelite will go into solution because the presence of calcium chloride, formed during the reaction, prevents it from dissolving. The same protective action of the calcium chloride appears to partly prevent the phosphorus from dissolving during the leach. It is then necessary to decant the calcium chloride solution and releach with fresh acid to remove the phosphorus. This is a critical step, because protective calcium chloride is not present and high losses of scheelite can occur, if the acid strength and contact time are not carefully controlled. The concentration of hydrochloric acid and the time required to dissolve the phosphorus without dissolving scheelite are determined in the laboratory before they are applied to plant practice. In plants where leaching is done in wooden tanks or in the presence of reducing agents such as carbon and organic matter, dissolved scheelite will reduce rapidly, imparting a telltale blue coloring to the solution. This gives the operator an indication that scheelite is being dissolved. However, the blue discoloration is not an entirely reliable indicator for loss of scheelite when leaching is done in the absence of a reducing agent. A loss of scheelite can occur undetected, except by assay.

The amount of acid required for leaching depends on the grade of concentrate and associated gangue. A small amount of acid is sufficient when the gangue is siliceous and the apatite content is low. Usually 12 to 25 pounds of commercial hydrochloric acid will dissolve up to 0.3 percent phosphorus. Concentrates produced from contact-metamorphic ores contain appreciable amount of calcite and require more acid.

Details of Leaching Procedure at Getchell

The flotation concentrate stored in the 44-foot thickener is pumped to the batch acid-leaching plant. Two to three tons of concentrate is charged to each of three 6- by 8-foot wooden agitator tanks. The charge is then settled and decanted; 20° B. commercial hydrochloric acid is added in agitation until effervescence ceases. The charge then is circulated through an acid-resisting, 36-inch, Fagergren flotation cell for the flotation of sulfide minerals and graphite. The circulation through the cell is continued until the froth is depleted and the scheelite is sulfide free. Reagents used for the sulfide float are American Cyanamid Co. Frother 77 and Palcotan. When flotation of the sulfides is completed, a specific gravity reading of the tailing is taken and the weight calculated. The charge is settled, and all of the solution (calcium chloride) is decanted, before leaching of the apatite. Water is added at the rate of 0.25 pound for each pound of concentrate and the charge agitated while 0.25 pound of hydrochloric acid for each pound of

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concentrate and the charge agitated while 0.25 pound of hydrochloric acid for each pound of concentrate is added to dissolve the apatite. The charge is agitated for one-half hour. The resulting leached concentrate (shown by a dotted line in the flowsheet) is filtered in a 42-inch, Shriver washing filter press. The acid solution is blown from the filter press. The cake is water-washed for one-half hour. The final concentrate is cleaned from the filter press, dried on electric driers, and sacked in 100-pound sacks. A comparison of the chemical analysis of the flotation concentrate and the final acid-leached concentrate is as follows:

	Flotation concentrate, percent	Acid-leached concentrate, percent
Tungstic trioxide.....	35.48	65.67
Phosphorus.....	2.19	.01
Sulfur.....	.40	.24
Insoluble.....	3.68	7.36

The success of leaching flotation concentrates into an acceptable grade depends on (1) a relatively high-grade flotation concentrate containing a minimum of acid insoluble material, (2) a closely controlled acid leach, and (3) flotation of sulfides in the acid circuit. This procedure does not remove molybdenum, except that which occurs as MoS_2, but it has permitted the Getchell mill to make a relatively high scheelite recovery in a concentrate that meets specifications.

Nevada-Massachusetts Co.

The mines and mill of the Nevada-Massachusetts Co. are at Tungsten, approximately 8 miles north of Mill City, Pershing County, Nev. The tactite ore, which contains approximately 0.4 percent WO_3, is concentrated by tabling, flotation, leaching, roasting, and magnetic separation. The mill treats 550 to 600 tons of ore in 24 hours, and produces a scheelite concentrate containing over 74 percent WO_3 at an average overall recovery of 80 percent. Figure 7 is the flowsheet of the milling operation.

Crushing and Grinding

Ore is trammed from the mine to the crushing plant by a gas-powered locomotive and dumped into two ore bins of 200 tons capacity each. Ore from the pit is truck-hauled. Chain feeders discharge the ore from the bins onto two 36-inch conveyor belts. These belts feed a 24- by 36-inch Allis-Chalmers blake-type crusher. The crusher discharges on a 24-inch conveyor belt that feeds a Stevens-Adams vibrating screen equipped with round steel screening rods having 3/4-inch openings. The screen oversize is conveyed to a 3-foot Traylor TY gyratory crusher. Its product joins the screen undersize material and is conveyed to a surge bin by a 24-inch conveyor. From the surge bin the product is carried and split at the discharge end of the conveyor to feed two 4- by 8-foot Symons screens equipped with round steel-screen rods that have 3/16-inch openings. The screen oversize product is fed by a 24-inch conveyor to a 4-foot shorthead Symons cone crusher, which is in closed circuit with the screens. All of the minus-3/16-inch product from the secondary screen passes

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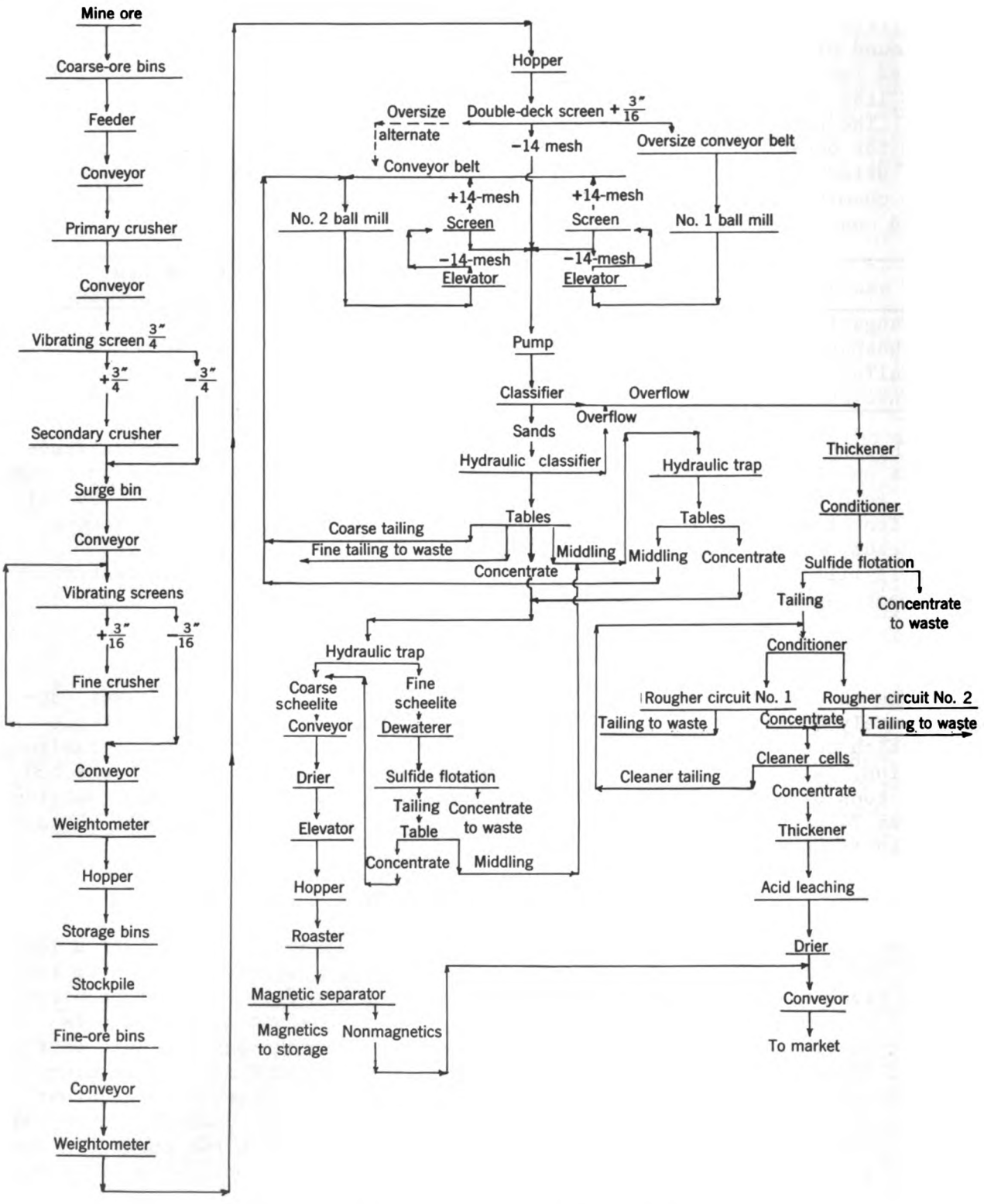

FIGURE 7. - Flowsheet, Nevada-Massachusetts Co.

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over a weightometer mounted on the conveyor belt and is discharged into a hopper. The hopper feeds a reversible, 16-inch, conveyor belt that discharges the product into the two end 300-ton storage bins; the hopper overflow feeds the center storage bin. The two end storage bins are equipped with short 36-inch belts; the center bin is equipped with a 24-inch belt to load haulage trucks that convey the material to an open stockpile near the mill, approximately 1,500 feet from the crushing plant. Under the stockpile is a concrete conveyor tunnel, housing a gate feeder and a 14-inch conveyor belt, which delivers the product to two 600-ton-each, circular, fine-ore bins. The products of the fine-ore bins are fed by Hardinge weightometers and are conveyed separately by 14- and 18-inch conveyors, respectively, to a hopper that feeds onto a double-deck scalping screen. The bottom deck is covered with a 14-mesh screen cloth. Oversize material from the top deck is conveyed to No. 1 ball mill; a portion of it is conveyed to No. 2 ball mill by the alternate conveyor shown in the flowsheet. Both ball mills are the March grate-discharge type.

The ball mills discharge into two wet-bucket elevators, where the products are elevated to two 14-mesh vibrating screens and wet-screened. The plus-14-mesh product from both screens goes to No. 2 ball mill for regrind, while the minus-14-mesh products join the minus-14-mesh product from the double-deck scalping screen and are pumped by a 4-inch Wilfley pump to an 8-foot Dorr classifier. Its overflow goes to flotation; its sands go to a 14-spigot Fahrenwald hydraulic classifier.

Gravity Concentration

The minus-14-mesh product to the Fahrenwald classifier is deslimed; the sands are classified into 14 separate spigot products. The gravity circuit consists of 16 tables - 6 Wilfley and 10 Deister-Overstrom diagonal. Each spigot product is fed separately to 14 tables; each table produces a concentrate, a middling, and a tailing. The first three tables treat the coarse spigot product. Tailings from these three tables are conveyed to ball mill 2 for regrind, while the finer tailings go to waste. Middling products from all 14 tables are separated into coarse and fine middling and the two sizes are treated separately on the remaining two tables. These two tables produce a concentrate and a middling. The middling is returned to ball mill 2 for regrind, while the concentrate from all 16 tables passes into a hydraulic trap to remove the coarse scheelite, which then goes to a drier. The fine scheelite is fed into a three-cell bank of Kraut flotation cells for a sulfide flotation. The sulfide concentrate is discarded, and the flotation tailing goes to a finishing table that produces a clean scheelite concentrate and a middling that is recirculated to the hydraulic trap. The clean scheelite concentrate goes by way of a screw conveyor into a drier and thence to a dry elevator, which discharges the product into a hopper feeding the roaster.

Roasting and Magnetic Separation

The roaster, locally designed and built, consists of an 8-inch steel pipe 12 feet long, inclined 1 inch per foot, brick-housed, and chain-driven at 15 r.p.m. It is externally heated with fuel oil to dull red heat. The material is passed through the roaster quickly to convert surfaces of the

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pyrite particles to magnetic iron sulfide. After cooling, the roasted material is passed over a three-pole crossbelt-type Dings Wetherill magnetic separator, where the magnetic pyrite is removed at the first two poles and the garnet at the third and most powerful pole. The magnetic iron product, which contains 6 to 7 percent WO_3, is stored for future treatment.

Flotation

The overflow from the Dorr and Fahrenwald classifiers, which are largely slimes, are pumped to a thickener and constitute an approximate 45 percent of the total mill feed. Occasionally a flocculent (Guartex - a General Mills product) is added to the thickener. The pulp from the thickener is pumped to a conditioner, where it is automatically regulated to a pH of 10.2 with soda ash. The pulp is conditioned for a sulfide float using xanthates, cresylic acid, and occasionally fuel oil. The sulfide float is made in a bank of eight 36-inch Agitair flotation cells. The sulfide concentrate is discarded, whereas the tailing goes to a second conditioner, where oleic acid, blown tall oil, sodium silicate, Palcotan or Quebracho, and sodium cyanide are added. The conditioned pulp is split to feed two rougher banks of eight 36-inch Agitair flotation cells. The rougher tailings go to waste. The concentrates go to a bank of six 30-inch Denver cleaner cells. Sodium silicate is added to the cleaner circuit. The rougher froth is pumped into No. 3 cleaner cell. Cleaner froth from No. 1 and No. 2 cells goes to No. 4 cell. Froths from No. 3 and No. 4 cells go to No. 5 cell, and the froth from No. 5 cell goes to No. 6 cell, where Palcotan is added for the final cleaning. The tailing from the cleaner circuit is returned to the conditioner, whereas the cleaned concentrate goes to a thickener. This thickness is used as a storage tank before acid treatment to eliminate calcite and apatite.

Leaching Operation

The concentrate from the thickener is pumped to two 3- by 3- by 15-foot, rectangular, rubber-lined, leaching vats, used alternately. Hydrochloric acid is added at the rate of 1 or 2 pounds per pound of concentrate, depending on the calcite content. The concentrate remains in the leaching vat until the calcite and apatite are dissolved, usually requiring several hours. The spent acid is removed from the leached pulp by decantation (siphoning), and the leached concentrate is given four water washes. The washed product is then hand-shoveled out of the leaching vat, dried in a rotary kiln, and blended with gravity concentrate.

Milling Data

Table 15 shows a screen analysis of composite samples of gravity and flotation mill feed and tailing, table 16 the yearly acid and flotation reagent consumption and table 17 the average yearly percentage of recovery, with a typical analysis of the tungsten concentrate.

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TABLE 15. - Screen analysis of composite samples, Nevada-Massachusetts Co., May 1955

Table tailing				Flotation tailing			
Mesh	Weight-percent	WO_3 percent	Percent of total WO_3	Mesh	Weight-percent	WO_3, percent	Percent of total WO_3
20	0.9	0.23	3.6	65	0.3	0.05	-
28	5.3	.07	6.4	100	1.1	.05	-
35	12.6	.06	13.0	150	3.9	.03	-
48	23.8	.06	24.6	200	8.9	.02	-
65	19.3	.04	13.3	-200	86.8	.11	-
100	18.4	.03	9.5				
150	11.2	.03	5.8				
200	5.7	.07	6.9				
-200	2.8	.35	16.9				
	Mill feed			Flotation feed			
1/4	14.7	0.17	7.5	65	0.1	0.07	-
10	48.0	.20	28.7	100	.8	.04	-
14	6.5	.38	7.4	150	3.1	.04	-
20	5.0	.39	5.8	200	7.2	.05	-
28	4.2	.50	6.3	-200	88.8	.39	-
35	2.7	.63	5.1				
48	2.5	.76	5.7				
65	1.9	.83	4.7				
100	2.0	.88	5.3				
-100	12.5	.63	23.5				

Product	WO_3, percent
Average mill feed.....	0.35-0.4
Average table tailing.....	.03- .04
Average flotation tailing...	.1
Concentrate grade.....	74+
Overall recovery.....	80.0+

Winnemucca Mountain Mining Co.

The mill of the Winnemucca Mountain Mining Co. is approximately 2-1/2 miles north of Winnemucca, Humboldt County, Nev. The mine is 42 miles west of Winnemucca. The ore (scheelite in quartz and pyrite) averages 0.71 percent WO_3. It is concentrated by tabling, followed by roasting and magnetic separation. The mill treats 1 ton of ore per hour at an average recovery of 62 percent. A 74 to 77 percent concentrate is produced and blended with the middling product for a final concentrate of 62 to 63 percent WO_3. Figure 8 is the flowsheet of the milling operation.

Crushing, Screening, and Grinding

The ore from the mine is truck-hauled to the mill and dumped on a grizzly with 5-inch openings. The grizzly oversize is fed to a Blake-type jaw crusher, and the crushed product joins the grizzly undersize. Then it is elevated to a vibrating screen having 5/8-inch openings. The screen oversize is fed to a

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No. 19-1/2 O.S. Kennedy Van Saun cone crusher, set at 1/4 inch and operated in open circuit. The crushed product joins the screen undersize and is conveyed to the fine-ore bin.

The fine-ore bin product is fed by a belt feeder to a 10-mesh trommel screen. The screen oversize is fed to a 30-inch Deco convertible grate discharge ball mill operated in closed circuit with a 10-mesh vibrating screen. All the combined minus-10-mesh products are fed to a 6- by 12-foot Wilfley table.

TABLE 16. - Reagent consumption, acid leaching and flotation

Year	Acid leaching, hydrochloric acid	
	Pounds per ton of ore	Pounds per unit WO_3 produced
1945.....	7.66	17.88
1946.....	8.87	23.90
1947.....	8.41	23.27
1948.....	6.17	15.00
1949.....	7.70	18.20
1950.....	6.42	17.42
1951.....	6.05	19.25
1952.....	5.74	20.37
1953.....	4.31	14.77
1954.....	5.77	18.66

Flotation-reagent consumption, pounds per ton of ore treated

Year	Soda ash	Oleic acid	Blown tall oil	Sodium silicate	Cresylic acid	Quebracho	Cyanide	R-301	Z-11
1945....	3.81	0.210	0.210	2.32	0.09	0.078	0.062	0.053	
1946....	4.01	.185	.211	1.64	.08	.051	.051	.042	
1947....	4.82	.098	.155	1.23	.08	.053	.046	.039	
1948....	4.36	.157	.168	2.06	.066	.142	.055	.048	
1949....	4.78	.164	.177	1.78	.067	.132	.048	.054	
1950....	4.75	.146	.161	1.85	.069	.15	.056	.052	
1951....	4.79	.117	.126	1.62	.072	.121	.048	.044	
1952....	4.69	.119	.127	1.53	.066	.111	.052	.044	
1953....	4.45	.102	.109	1.44	.082	.115	.071	.064	
1954....	5.14	.091	.099	1.19	.078	.096	.063	-	0.049
Average reagent consumption, pounds per ton of flotation feed treated									
	10.75	0.19	0.207	2.50	0.165	0.201	0.131	-	0.102

Table Concentration

No attempt is made to deslime or to classify the sand product because of the shortage of water and the cost of reclaiming it from the tailing pond. The minus-10-mesh product is fed directly to the Wilfley table at the rate of 1 ton per hour, producing a concentrate, a middling, and a tailing. The tailing, containing 0.18 percent WO_3, is discarded; but the concentrate and middling product that contain appreciable pyrite are kept separate for roasting and magnetic separation. No attempt is made to regrind the middling product, which is roasted to produce a 28-percent WO_3 product by magnetic separation;

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this product, when blended with the 74 to 77 percent concentrate, results in a marketable finished concentrate containing 62 to 63 percent WO_3.

TABLE 17. - Average yearly recovery and typical concentrate analysis

Year	Table-concentrate recovery, percent	Flotation-concentrate recovery, percent	Total recovery, percent
1945.....	80.52	72.61	75.84
1946.....	84.63	67.87	79.00
1947.....	84.60	70.06	77.01
1948.....	80.66	75.28	77.73
1949.....	81.00	75.10	77.84
1950.....	81.67	79.33	80.45
1951.....	66.09	78.39	73.09
1952.....	82.04	77.94	79.84
1953.....	87.43	74.51	81.41
1954.....	87.86	69.69	79.86

Typical analysis of tungsten concentrate

Concentrate	Analysis, percent	Concentrate	Analysis, percent
Tungsten trioxide....	74.32	Molybdenum.....	0.36
Tin.....	.01	Manganese.....	.05
Arsenic.....	.05	Bismuth.....	.04
Sulfur.....	.33	Lead.....	.04
Copper.....	.02	Zinc.....	.04
Total phosphorus.....	.02	Iron.....	.83
Antimony.....	< .01	Moisture.....	.052

Roasting and Magnetic Separation

The concentrate and middling are roasted separately in a steel tube 12 inches in diameter and 6 feet in length, externally heated with butane bottle gas. Gas consumption averages 28 gallons per ton of concentrate treated at a cost of 18 cents per gallon. The tube furnace is rotated at 3 r.p.m. and heated to 600° C. at the hot zone (discharge end). The material remains in the hot zone only a few seconds. After cooling the product is magnetically separated with a Dings 12- by 12-inch permanent magnetic separator, and the two nonmagnetic products are blended to produce the desired grade of concentrate, and the magnetic product (which contains 3 percent WO_3) is stored for future treatment.

The Wolfram Co.

The Toulon mill of the Wolfram Co. is approximately 15 miles south of Lovelock on Highway 40, Pershing County, Nev. The ore is truck-hauled to the mill from the company Nightingale mine. Occasionally custom ores are treated at the mill. The ore from the Nightingale mine is principally tactite, containing scheelite, garnet, apatite, epidote, calcite, pyrite, pyrrhotite, quartz, and small amounts of powellite. The tungsten mineral is concentrated by gravity methods, followed by sulfide flotation of the gravity-concentrate product and roasting and magnetic separation of the sulfide-flotation tailing.

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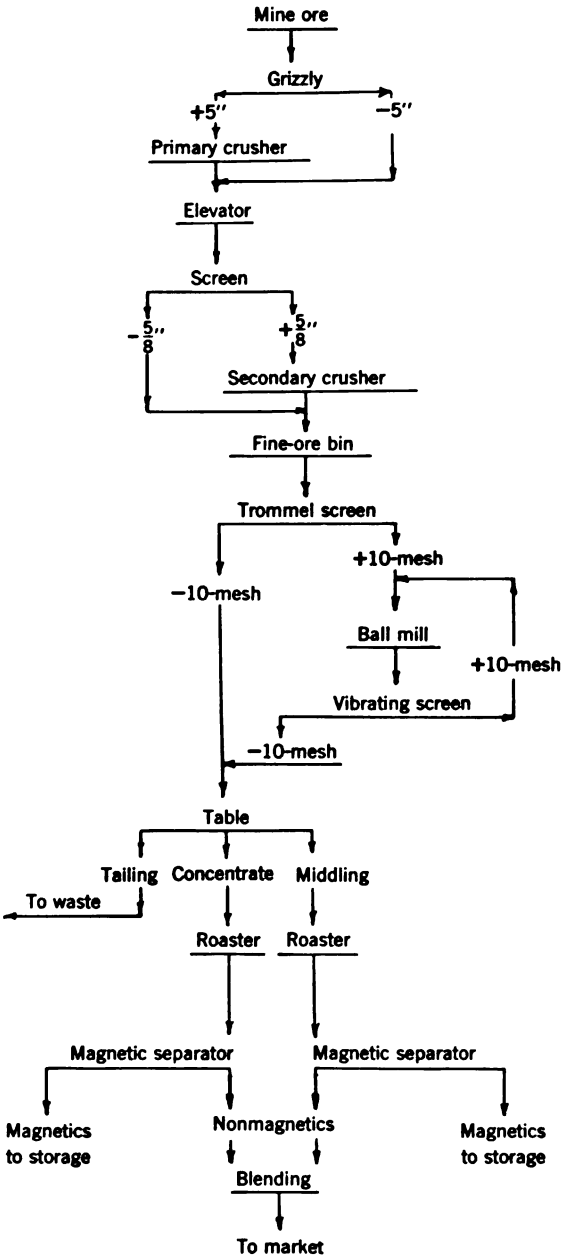

FIGURE 8. - Flowsheet, Winnemucca Mountain Mining Co.

The mill treats 120 tons of ore per 24 hours, yielding a final product that assays 63 to 78 percent WO_3. The average mill feed assays 0.4 percent WO_3 and tailing 0.15 percent WO_3; and the recovery is 72 to 74 percent. Figure 9 is the flowsheet of the milling operation.

General Milling Practice

The ore from the mine is truck-hauled to a platform from whence a bulldozer scrapes it into the coarse-ore bin. A pan feeder feeds the ore to a 10- by 20-inch Farrell jaw crusher for crushing to 2-1/2 inches. A conveyor delivers the crushed product to an elevator, which discharges into a 9/16-inch-opening trommel screen. The oversize product goes to a secondary, 24-inch, Telsmith gyratory crusher in closed circuit with the elevator and screen. The undersize product goes to the fine-ore bin. A belt conveyor delivers the ore from the fine-ore bin to another elevator, which discharges it to a Hum-mer double-deck vibrating screen. The top deck is equipped with an 1/8-inch opening Tyrod No. 9633 screen cloth and the lower deck with a 16-mesh Tyler or Tyrod screen cloth. The oversize from the top deck goes to No. 1 rolls, and the oversize from the lower deck to No. 2 rolls. Both rolls are in closed circuit with the screen. The undersize from the lower deck screen go to a 24- by 24-inch Bendelari jig. The jig acts more as a sizer than a concentrator, producing a tailing and a hutch product. The gravity circuit consists of seven Wilfley tables, but only six are in use. The jig hutch product is fed to tables 1 and 2, and the tailing to a four-spigot Fahrenwald hydraulic classifier. Spigot 1 is not being used at present. Spigot products 2 and 3 are combined and fed to table 3. Spigot product 4 is fed to table 4. All the tables produce a concentrate, a middling, and a tailing.

The hydraulic classifier overflow, which constitutes fines and slime, flows to a thickener, from whence a 2-inch Dorrco diaphragm pump delivers the pulp to table 7. Middling products from all the tables are combined and also are fed to table 7. The middling from table 7 goes to an elevator, which

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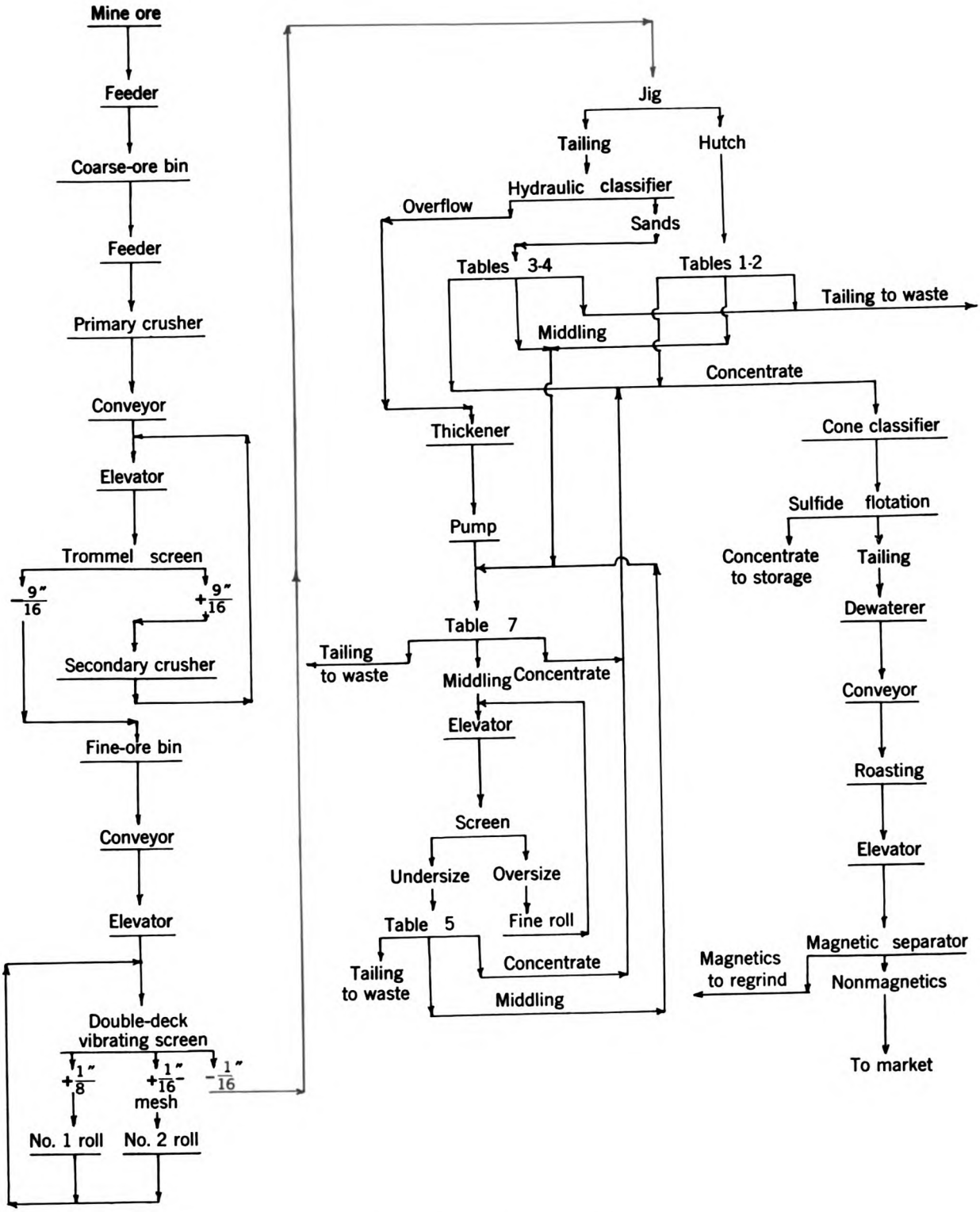

FIGURE 9. - Flowsheet, Toulon Mill, Wolfram Co.

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discharges onto a screen. The screen oversize goes to a fine roll, in closed circuit with the elevator. The undersize product is fed to table 5; the middling from table 5 returns to table 7. The tailing from all six tables is discarded to waste, and the concentrate from all six tables is combined for the sulfide flotation.

Flotation of sulfide minerals is made in two 24- by 24-inch Galigher cells. The reagents used are xanthate 301 and a frother 242 mixture composed of one part of frother 242 and four parts of cresylic acid. The sulfide concentrate is stored for future treatment. The flotation tailing is dewatered in a screw conveyor and delivered to the roaster for drying and roasting. The roasted product goes to an elevator, which feeds a four-belt Wetherill-type magnetic separator. The magnetic products contain pyrite and garnet. The pyrite is discarded, and the garnet is returned to regrind for reuse in tabling operations. Because it is difficult to distinguish between scheelite and quartz in table operations, the garnet is used as an indicator in making a close split between these minerals to produce a high-grade tungsten concentrate. The nonmagnetic product is sacked for shipment.

Nevada Scheelite Co.

Division of Kennametal, Inc.

Nevada Scheelite Co., a division of Kennametal, Inc., is approximately 5 miles east of Rawhide, Mineral County, Nev. The ore, containing scheelite, is mainly a porous limonitic tactite made up of garnet, epidote, apatite, diopside, limonite, pyrite, calcite, wollastonite, and quartz. The scheelite is concentrated by tabling and flotation, followed by roasting and magnetic separation to remove sulfides and garnet. A two-stage acid leaching of the flotation concentrate removes calcite and apatite. The mill treats 125 tons of ore per 24 hours. The average mill feed assays 1.2 percent WO_3, the average tailing 0.17 percent WO_3, and the concentrate 70.0 percent WO_3 at an overall recovery of 85 percent. Figure 10 shows the flowsheet of the milling operations.

Crushing and Grinding

The ore from the mine is hoisted in a 2-ton skip and dumped into a 100-ton steel coarse-ore bin, from whence an apron feeder delivers the ore to a 15- by 24-inch Universal jaw crusher, where it is crushed to a 3-inch product and conveyed to a 20-ton crushed-ore bin. A 4- by 7- by 22-inch D.P. magnetic separator is suspended over the conveyor belt to remove tramp iron and occasional pieces of coarse magnetite ore. The crushed ore is then fed to a No. 25, Kennedy, gearless, gyratory crusher, reduced to 1/2 inch, and conveyed to an 80-ton fine-ore bin. A Syntrol electric vibrator feeds the ore from the fine-ore bin onto a conveyor belt, which delivers the product to a bucket elevator. The elevator discharges to a 4- by 6-foot, Overstrom, double-deck step screen. The top deck is equipped with a 10-mesh screen cloth and the lower deck with a 20-mesh screen cloth. The top deck acts as a scalper to relieve the load on the lower deck. The plus-10-mesh and plus-20-mesh products are combined and fed to a 5- by 10-foot Straub, double-compartment, peripheral-discharge ball mill fed from both ends. The ball mill is in closed

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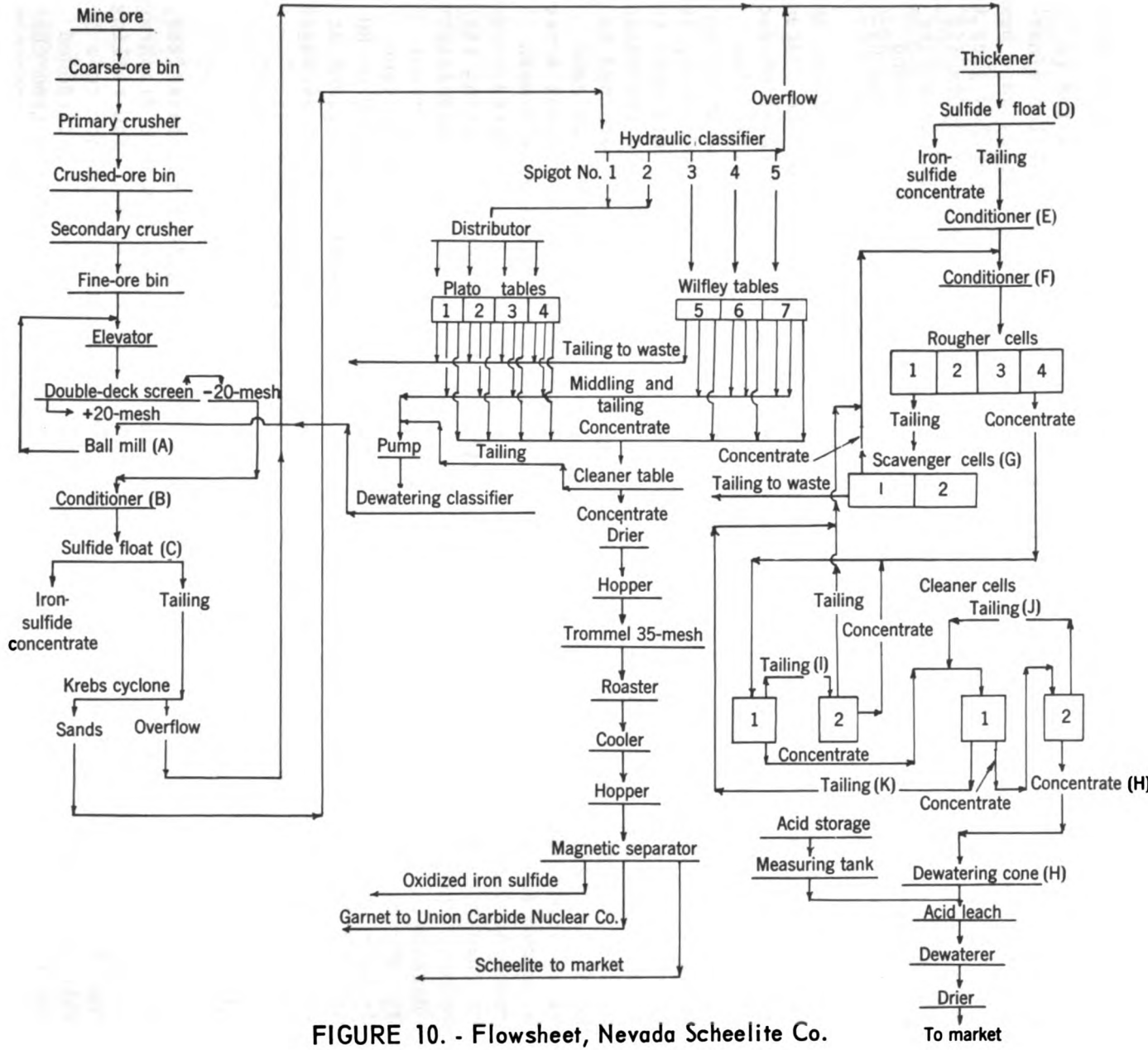

FIGURE 10. - Flowsheet, Nevada Scheelite Co.

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circuit with the elevator and screen. The minus-20-mesh product goes to a 5- by 5-foot Wemco conditioner in preparation for the sulfide flotation.

Sulfide Flotation, Gravity Concentration, Roasting, and Magnetic Separation

The conditioned minus-20-mesh pulp in the Wemco conditioner is fed to three 44- by 44-inch Booth flotation cells for removing pyrite sulfide minerals. The tailing from the flotation cells flows to a 2-inch Wilfley sand pump and then is pumped to a Krebs cyclone. The cyclone slimes or overflow goes to an 8- by 24-foot Dorrco thickener and the sands to a 5-pot Deister hydraulic classifier. A 2-inch Denver vertical pump delivers the hydraulic classifier overflow to the 8- by 24-foot Dorrco thickener in preparation for a second flotation of the sulfide minerals. Sands from the first two pots of the hydraulic classifier are combined and pumped by a 2-inch Wilfley pump to a Denver self-rotating distributor that distributes the feed to three Plato, and one Diester diagonal table. The remaining three pot products from the hydraulic classifier are fed separately to three Wilfley tables. Each table produces a concentrate, a middling, and a tailing. The tailing from the first five tables goes to waste. The tailing from the last two tables and the middling from all the tables are pumped by a 2-inch Wemco sand pump to a 30-inch by 16-foot Wemco screw classifier for dewatering, thence to the Straub ball mill to be reground. The concentrate from all seven tables is combined and recleaned on a Diester diagonal cleaner table. Tailing from this table is returned to the Straub ball mill for regrinding; the concentrate goes to a Sutton sand drier, whence it is carried to a hopper feeding a small 35-mesh trommel screen to remove any plus-35-mesh product that may be present. The minus-35-mesh screen product is fed to an externally fired roaster tube. The roasted product then goes to a screw feeder equipped with a water jacket for cooling the product, then to a hopper that feeds a Wetherill-type 3 cross belt magnetic separator. The oxidized iron sulfide containing approximately 1.5 percent WO_3 is removed at the first crossbelt. The garnet containing 15.0 percent WO_3 is removed at the second crossbelt. This product, which totals 60 or 80 units of WO_3 per month, is shipped to the Union Carbide Nuclear Co. plant at Pine Creek, Calif., for further processing. The nonmagnetic product averages 70.0 percent WO_3 and is sacked in 100-pound bags for shipment.

Flotation

The slime overflow pulp from the Krebs cyclone and hydraulic classifier, after thickening, is pumped with a 3-inch, Simplex, low-head, Denver diaphragm pump to a No. 250 Denver unit cell for further pyrite flotation. The tailing from the unit cell goes to a 6- by 6-foot redwood conditioner, thence to a 5- by 5-foot redwood conditioner before fatty acid flotation. The conditioned pulp flows to four 36- by 36-inch Booth rougher cells. The tailing from the rougher circuit goes to two Booth scavenger cells before final disposal; the scavenger concentrate is returned to the conditioner. The rougher concentrate goes to a bank of four Denver cleaners that consists of two No. 12 and two No. 10 Denver cells for a three-stage cleaning. The cleaner tailing is returned to the conditioner. The finished concentrate is dewatered in a dewatering cone in preparation for the acid treatment.

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Acid Treatment

The acid-treatment plant consists of a 4- by 4- by 15-foot rubber-lined treatment tank, a 4- by 4-foot, rubber-lined acid-measuring tank, a 10,000-gallon, rubber-lined, acid-storage tank, and a locally designed gas-fired drier having brick sides and a heavy top steel plate. Two-stage leaching is practiced. Hydrochloric acid consumption varies from 100 to 200 pounds per unit of tungstic trioxide treated.

In the first stage of leaching a 3-hour cycle is used to dissolve the calcite. The resulting calcium chloride solution is siphoned off and discarded. In the second stage of leaching a 45-minute agitation in 20-percent hydrochloric acid solution is used to dissolve the apatite, followed by two water washes. The final leached product is dried and sacked in 100-pound bags for shipment. The milling data are summarized below:

Product	WO_3, percent
Average mill feed	1.2
Average flotation feed	.65
Average table feed	1.45
Average flotation tailing	.20
Average table tailing	.14
Average recovery, flotation	65.0-68.0
Average recovery, tabling	88.0-90.0
Overall recovery	83.0
Average grade, gravity concentrate	70.0+
Average flotation concentrate, untreated	25.0
Average flotation concentrate, acid-treated	68.0

Typical analysis of finished combined gravity and flotation concentrates:

	Percent
Tungsten trioxide	69.52
Sulfur	.13
Phosphorus	.01
Copper	.03

Table 18 shows the reagent consumption and place of addition and table 19 a typical screen analysis.

Dead Horse Wells Unit

The Dead Horse Wells mill, a unit of the Nevada Scheelite Co., is 8 miles southwest of Rawhide, Mineral County, Nev. In 1956 the mill treated tailing from past operations of the Nevada Scheelite mine. The original tailing pile contained approximately 55,000 tons of which 27,000 tons still remain to be treated.

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TABLE 18. - Flotation-reagent consumption and place of addition, Nevada Scheelite Co.

Reagent	Pounds per ton	Place of addition	Letters shown in figure
Soda ash	1.50	Ball mill	(A)
Reagent Z-6 + Z-11	.15	Conditioner	(B)
Pine oil + frother-77 (1-1)	.15	Sulfide flotation	(C)
Frother-77 + R-Z-11 each	.05	do.	(D)
Soda ash	15.00	Conditioner	(E)
Sodium silicate	2.50	do.	
Sodium cyanide	.30	do.	
Palcotan	.50	do.	(F)
Oleic acid + blown tall oil (1-1)	.60	do.	
Aerosol 18	.05	do.	
Oleic acid + blown tall oil (1-1)	.10	Scavenger circuit	(G)
Alum sulfate	.05	Dewatering cone	(H)
Sodium silicate	1.00	Cleaner circuit	(I)
Aerosol 18	.01	do.	(J)
Sodium cyanide	.10	do.	(K)

TABLE 19. - Typical screen analysis, Nevada Scheelite Co.

	Flotation feed			Flotation tailing			Table tailing		
Mesh	Weight-percent	WO_3 per-cent	WO_3 distribution, percent	Weight-percent	WO_3 per-cent	WO_3 distribution, percent	Weight-percent	WO_3, per-cent	WO_3 distribution, percent
+ 35	-	-	-	-	-	-	22.1	0.17	24.5
+ 48	-	-	-	-	-	-	29.1	.14	26.7
+ 65	3.1	0.50	2.6	4.9	0.37	11.2	25.2	.15	24.7
+100	4.7	.51	4.3	5.3	.27	8.8	14.5	.18	17.0
+150	7.8	.36	4.9	7.2	.13	5.6	5.0	.16	5.2
+200	17.2	.33	10.0	14.6	.09	8.2	1.5	.14	1.4
-200	-	-	-	-	-	-	2.6	.03	.5
+270	13.7	.38	9.2	11.0	.08	5.6	-	-	-
-270	53.5	.73	69.0	57.0	.17	60.6	-	-	-
Calculated feed		.57		Calculated tailing		.16	Calculated tailing		.15

Tungsten minerals found in the tailing are complex. They possibly contain reinite or what is known locally as "spookite." Because of their complexity, recovery is low. The mill heads assay 0.45 percent WO_3 and the tailing 0.33 to 0.34 percent WO_3, resulting in an average recovery of 25 percent. The average grade of the flotation concentrate is 12.0 percent WO_3; when acid-treated, it is upgraded to 62.0 to 63.0 percent WO_3. The mill treats 125 tons of tailings per 24 hours. Figure 11 is the flowsheet of the milling operation.

General Milling Practice

A rotary scraper picks up the tailing and feeds it to a 10-ton hopper, whence a pan feeder feeds the material to a conveyor that discharges to a

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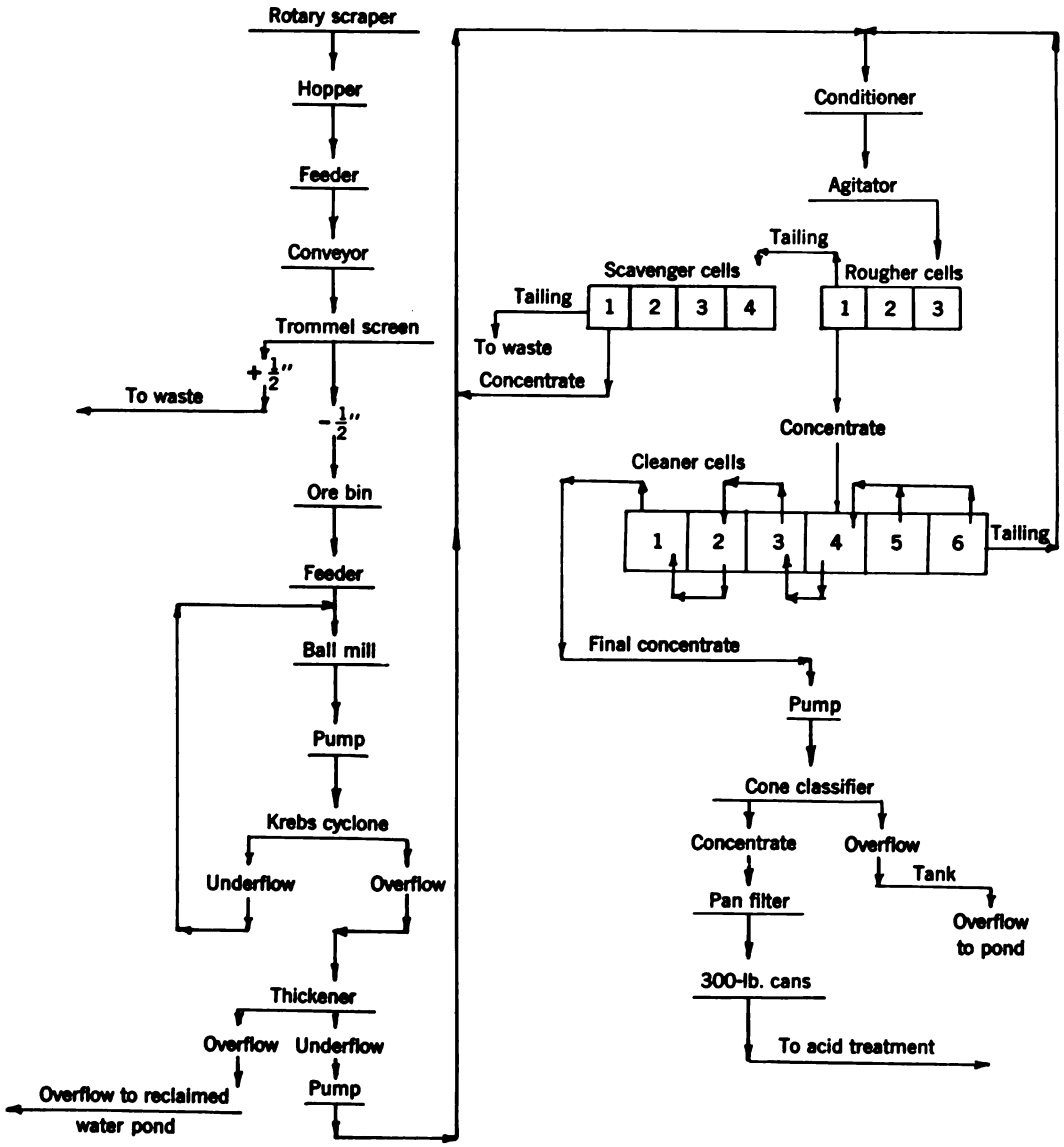

FIGURE 11. - Flowsheet, Dead Horse Wells Unit, Nevada Scheelite Co.

1/2-inch-opening trommel screen. The screen oversize is discarded to waste, and the undersize is discharged into a 150-ton mill ore bin. A belt feeder delivers the product from the mill ore bin to a conveyor, which discharges to a 5- by 5-foot Marcy ball mill. The ball mill discharges to a 3-inch SRL pump. The product is pumped to a 10-inch Krebs-cyclone classifier, which is in closed circuit with the ball mill. The Krebs-cyclone classifier overflow moves by gravity to a 34- by 10-foot thickener. The thickener overflow goes to the reclaimed water pond, and the underflow is pumped with a 4-inch Denver diaphragm pump to a 6- by 7-foot conditioner and thence to a 5- by 5-foot agitator. The conditioned pulp is fed to three 44-inch Fagergren rougher cells. The rougher concentrate goes to the cleaner circuit and the tailing to four 36-inch Fagergren scavenger cells. The concentrate from the scavenger

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circuit goes to a 1-1/2-inch Denver vertical pump and is pumped to the No. 1 conditioner. The scavenger tailing goes to a 3-inch Wilfley pump and is pumped to waste.

The cleaner circuit consists of six Denver cleaner cells. The rougher concentrate is fed into cell 4 and its concentrate passes through a three-stage cleaning. The cleaner tail joins the scavenger concentrate and is pumped to the conditioner. The final concentrate flows to a 1-inch Denver vertical pump and is pumped to a cone classifier. The overflow from the settling cone goes to a rectangular redwood settling tank and thence to the reclaimed water pond. The concentrate underflow of the cone classifier goes to a 3- by 3-foot Denver pan filter and thence to cans holding 300 pounds. The air-dried concentrate is then trucked 16 miles to the Nevada Scheelite plant for acid treatment. Table 20 gives the consumption and places of addition of the flotation reagents.

TABLE 20. - Flotation-reagent consumption and place of addition, Dead Horse Wells unit

Reagent	Pounds per ton	Place of addition
Soda ash	6.00	Ball mill
Sodium silicate	4.50	Do.
Sodium cyanide	.50	Do.
Soda ash	12.00	Conditioner
Oleic acid-blown tall oil (2-1)	.40	Do.
Aerosol	.05	Do.
Palcotan	.20	Do.
S-541 frother	.20	Rougher
S-541 frother	.30	Scavenger
Oleic acid-blown tall oil (2-1)	.15	Do.
Palcotan	.05	1st cleaner
Sodium silicate	.50	Recleaner

Gabbs Exploration Co.

The mill of the Gabbs Exploration Co. is one-half mile west of Gabbs, Nye County, Nev. The ore from the company Victory mine, approximately 10 miles north of Gabbs, is truck-hauled to the mill. The ore is tactite containing scheelite, calcite, garnet, pyrite, quartz, and minor amounts of apatite and epidote. The scheelite is concentrated by gravity and flotation methods, followed by two-stage leaching, roasting, and magnetic separation of the scheelite concentrate product. The mill treats 80 tons of ore per 24 hours. The average mill feed contains 1.5 percent WO_3, and tailing contains 0.15 percent WO_3, a recovery of 92 percent or more. The grade of blended gravity and flotation concentrate averages 64 percent WO_3. Figure 12 is the flowsheet of the milling operation.

Crushing and Grinding

The ore at the mine passes through an 8-inch grizzly into an 80-ton ore bin, from which it is trucked to the mill's 25-ton coarse-ore bin. An

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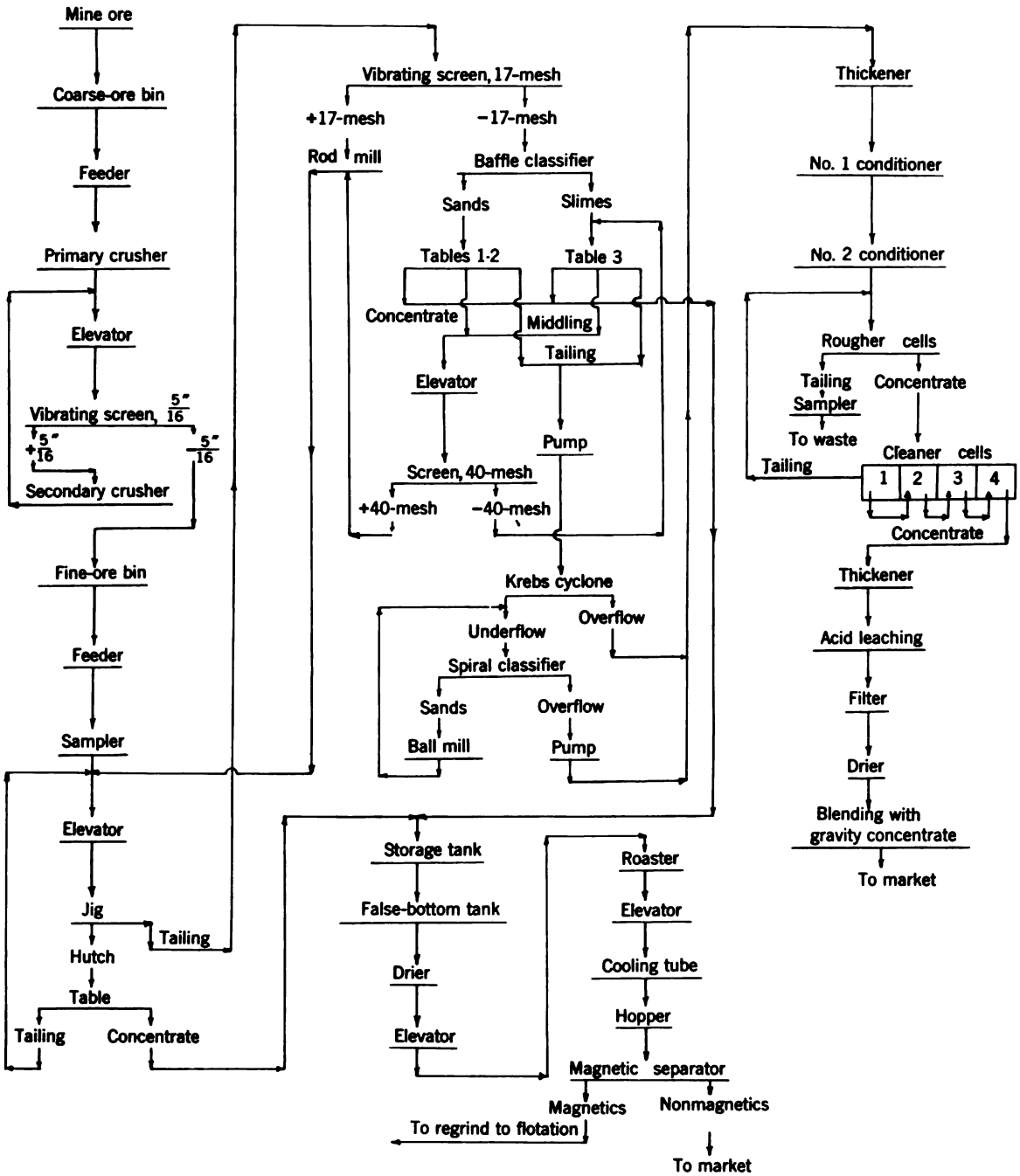

FIGURE 12. - Flowsheet, Gabbs Exploration Co.

apron feeder conveys the ore from the coarse-ore bin to a 20-inch Kue-Ken jaw crusher for reduction to 1 inch. The crushed product is elevated to a 5/16-inch-opening Concenco vibrating screen. The screen oversize goes to an 18-inch Kue-Ken gyratory crusher, which is in closed circuit with the elevator and vibrating screen. The screen undersize goes to the mill 100-ton fine-ore bin.

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A 12-inch belt feeder at the fine-ore bin delivers the ore to an elevator. An automatic sampler samples the ore in transit to the elevator. The elevator discharges the ore to a 24-inch Bendelari jig. The jig produces a tailing and a hutch product. The jig tailing is fed to a 17-mesh Selectro vibrating screen. The screen oversize goes to a 3- by 8-foot Wemco rod mill, in closed circuit with the elevator and jig; the undersize goes to two Deister cone baffle classifiers. The jig hutch product goes to a half-size Deister, diagonal-deck shaking table. The table concentrate joins other table concentrates, and the table tailing is returned to the elevator feeding the jig.

Table Concentration

The two spigot products from the two Deister cone baffle classifiers are fed to the Deister Plato tables 1 and 2. The slime overflow from the cone classifier is fed to table 3. Each table produces a concentrate, a middling, and a tailing. The combined table concentrates flow to a 1-inch Denver vertical pump and are pumped to a storage tank in preparation for acid treatment. The combined table middlings go to an elevator that discharges to a 40-mesh screen. The minus-40-mesh product is fed to table 3. The plus-40-mesh product is returned to the rod mill for regrind. The combined table tailings flow to a 3-inch SRL pump and are pumped to a 10-inch Krebs cyclone classifier. The cone-classifier underflow goes to a 24-inch Wemco spiral classifier, which is in closed circuit with a 3- by 8-foot Wemco ball mill, which grinds the product to 48-mesh. The classifier overflow from the spiral classifier flows to a 2-inch Wemco pump and is pumped to a 16-foot thickener in preparation for flotation. The cyclone overflow flows by gravity to the same thickener.

Flotation

The thickener overflow is reclaimed water.

A 3-inch Dorrco duplex diaphragm pump raises the thickener underflow to No. 1 conditioner, from whence it flows by gravity to conditioner 2. Both conditioners are 3-1/2 feet square and 4 feet deep.

The conditioned pulp in conditioner 2 is pumped with a 1-1/2-inch Wemco pump to four 44-inch Fagergren rougher cells. The last two rougher cells can be used as scavengers, if necessary. The rougher tailing is sampled with a Galigher sampler, then flows through a 3-inch Wemco pump to waste. The rougher concentrate flows to four No. 12 Denver cleaner cells for a three-stage cleaning. The cleaner tails are returned to the head of the rougher circuit, and the final concentrate goes to a 12-foot Wemco thickener in preparation for acid treatment.

Acid Treatment

Flotation Concentrates. - The final cleaner concentrate contained in the thickener is distributed to two 5- by 5-1/2-foot redwood, acid-leaching tanks equipped with rubber-covered impellers. A two-stage leaching is employed to dissolve the calcite and apatite separately. In the first leaching stage hydrochloric acid is stage-fed until effervescence ceases. The pulp is then

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allowed to settle, and the resulting calcium chloride solution is removed by decantation. In the second leaching stage 8-percent hydrochloric acid is added for 1-1/2 hours. The dissolved apatite is then removed by decantation, and the remaining pulp is given two water washes by decantations. The leached and washed scheelite concentrate goes to a 4-foot pan filter, then to a drier. The dried product is mixed with gravity concentrates to produce a 64-percent WO_3 product.

Gravity Concentrates. - The combined gravity concentrates in the storage tank, previously mentioned, go to a 3-1/2- by 3-1/2-foot, redwood, false-bottom tank. Eight percent hydrochloric acid is added and the concentrate leached by percolation for 1-1/2 hours to remove apatite. The leached and washed product then goes to a sand drier in preparation for roasting and magnetic separation.

Roasting and Magnetic Separation

The product from the sand drier goes to an elevator that discharges the product to an 8- by 12-foot, gas-fired, stainless steel pipe roaster. The roasted product is elevated to a cooling tube, and the cooled product is discharged to a hopper feeding the Stolle magnetic separator. The magnetic product is returned to regrind and flotation. The nonmagnetic product is blended with flotation scheelite concentrate for sacking and shipment. Table 21 gives the flotation-reagent consumption and place of addition.

TABLE 21. - Flotation-reagent consumption and place of addition, Gabbs Exploration Co.

Reagent	Pounds per ton	Place of addition
Soda ash.....	6-10	Ball mill
Sodium silicate.....	4.00	Do.
Palcotan.....	.20	Do.
Sodium cyanide.....	.1-0.2	Do.
Oleic acid.....	.30	Conditioners 1, 2
Cyanamid 710.....	.20	Do.
Aerosol 18.....	.15	Rougher
Sodium silicate.....	.20	Cleaner
Palcotan.....	.30	Do.

Yaney Milling Co.

The Yaney Milling Co., a custom tungsten-milling organization, is 2 miles northeast of Gabbs, Nye County, Nev. Custom ores from various sources are truck-hauled to the mill. The texture of custom ores, as well as the grade, varies considerably. Generally speaking, the ores are tactite containing scheelite, calcite, epidote, apatite, pyrite, pyrrhotite, quartz, dolomite, manganese minerals, and occasionally copper minerals. The grade of the custom ores ranges from 0.4 to 2.5 percent WO_3. The scheelite is concentrated by gravity and flotation methods. The flotation concentrate is treated by a two-stage leaching to remove calcite and apatite. The gravity concentrate is treated by magnetic separation. The mill treats 65 tons of ore per 24 hours,

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producing a gravity concentrate that averages 70-percent WO_3 and a flotation concentrate that assays 20 to 30 percent WO_3, which, when acid treated, is upgraded to between 50 and 65 percent WO_3. The tailing assay on a 1.0-percent feed is usually 0.12 percent WO_3. The acid-treated flotation concentrate is mixed with gravity concentrates to yield a marketable product. Figure 13 is the flowsheet of the milling operations.

Crushing and Grinding

Custom ores are truck-hauled to the mill's coarse-ore bin, from which the ore is fed by means of a chute feeder to an 8- by 16-inch Wheeler jaw crusher, which crushes the ore to 1-1/2 inches. A conveyor belt delivers the crushed product to a 2-foot Traylor gyratory crusher for the secondary crushing. The Traylor crusher product is discharged into an elevator that distributes the ore to either of two 65-ton-capacity circular fine-ore bins. A belt feeder delivers the ore from the fine-ore bin to a wet elevator, which discharges to a 12- by 18-inch, two-compartment Denver duplex jig. The jig tailings go to a 2- by 5-foot Selectro 20-mesh screen, and the jig hutch product goes to concentrating table 3. The screen oversize goes to a 52- by 47-inch ball mill and the screen undersize to a cone classifier in preparation for tabling. The ball mill discharges into a rake classifier. The classifier sands are returned to the wet elevator, and the classifier overflow, containing solids between 60- and 80-mesh goes to the flotation conditioner.

Table Concentration

All the tables are Deister diagonal tables. The cone sands from the screen undersize are fed to table 1, the cone overflow to table 2. The middlings from tables 1 and 2 are fed to table 3, together with the jig hutch product. The combined concentrates from all three tables go to an Esperanza dewaterer and then to a rotary drier and magnetic separator.

The tailings from all three tables and the middlings from table 3 are combined and flow to a cone classifier. The cone underflow sand flows to the ball mill for regrind, while the cone overflow goes to a 20-foot thickener in preparation for flotation.

Flotation

The thickener overflow is used as reclaimed water, and the underflow goes to a 5- by 6-foot conditioner, where appropriate reagents are added. The conditioned pulp flows to eight No. 24 Denver rougher cells. The rougher tailing flows to waste and the rougher concentrate goes to a bank of six Denver Sub-A cleaner cells for a three-stage cleaning. The cleaner tailing is returned to the head of the rougher circuit and the final cleaner concentrate to the acid-leaching tanks.

Acid Leaching

The acid-leaching tanks consist of three 4-1/2- by 7-foot redwood tanks. Two-stage leaching is employed. In the first leaching stage, hydrochloric

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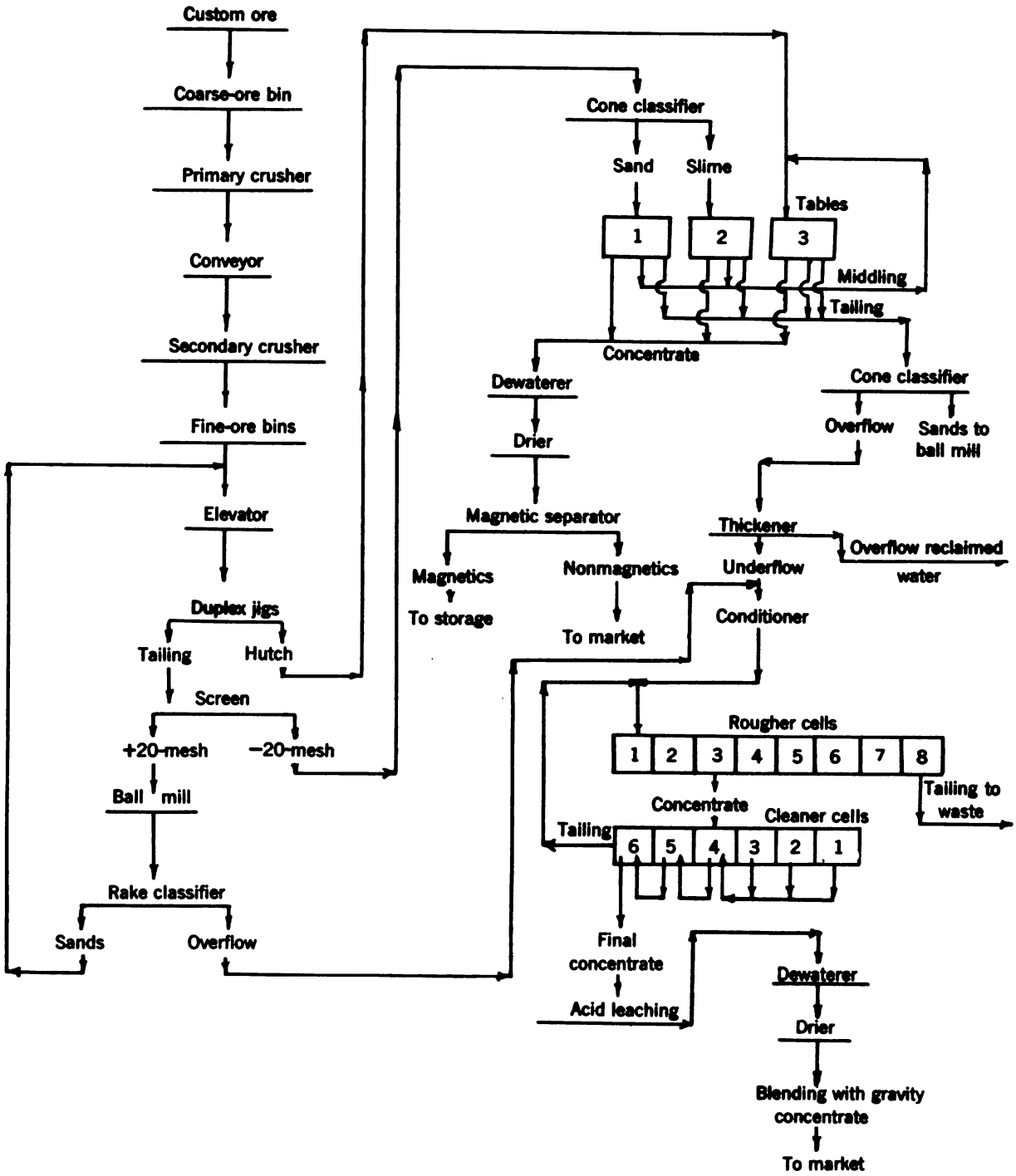

FIGURE 13. - Flowsheet, Yaney Milling Co.

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acid is added to dissolve the calcite. The resulting solution is discarded by decantation. In the second leaching stage 15 percent (9.8° B.) hydrochloric acid is added to dissolve the apatite. The acid is permitted to remain in contact with the pulp until a blue coloration appears, which indicates that the apatite is dissolved and the scheelite is being attacked. At this point the solution is discarded by decantation. The remaining pulp is given four water washes. The scheelite product is then dewatered, dried, and blended with gravity concentrates to produce a marketable product.

Magnetic Separation

The magnetic separator is a locally built five-belt Wetherill type. The magnetic products consist of iron sulfide minerals and garnet. The sulfides are discarded, and the garnet is stored for use in the gravity circuit as an indicator in tabling when custom ores do not contain garnet. The presence of garnet permits a closer cut on the tables between scheelite and quartz, which results in a higher recovery and a higher grade scheelite concentrate. The nonmagnetic product is sacked for shipment.

Minada Corp.

The Minada mill of the Minada Corp., formerly known as the Nevada Tungsten Corp., is at Sodaville, 3 miles south of Mina on Highway 95, Mineral County, Nev. Ore from the company Silver-Dyke mine is truck-hauled to the mill. The mine ore averages 1.0 percent WO_3 and is characterized as quartz containing scheelite, calcite, epidote, apatite, garnet, and pyrite. The mill is an all-gravity circuit that has a capacity between 125 and 150 tons of ore per 24 hours. The Silver Dyke mine produces 50 to 75 tons of ore per day. With the present gravity facilities, 80 to 85 percent recovery is being attained in milling the mine ore. Figure 14 is the flowsheet of the milling operations.

Crushing and Grinding

The ore from the mine is truck-hauled to the mill site and dumped into a 12- by 12-foot coarse ore bin and fed to a 15- by 28-inch Wheeler jaw crusher. The crushed product is conveyed to a 3- by 6-foot Denver Dillon vibrating screen having 3/8-inch openings. The screen oversize is fed to a 32-inch Telesmith crusher; the screen undersize joins the crushed product, is sampled, and is conveyed to another 24- by 48-inch vibrating screen also having 3/8-inch openings. The plus-3/8-inch product is routed to bin 1, and the minus-3/8-inch product to bin 2. Each bin is equipped with individual feeders. The product from bin 1 is fed to a 6- by 4-foot Marcy ball mill that is in closed circuit with a 22-inch by 6-foot Duplex rake classifier. The classifier overflow flows to a 2-inch Wilfley pump and is pumped to a two-cell, 18- by 24-inch Denver jig. The product from bin 2 is fed directly to the same jig. The jig tailing flows to a 20- by 60-inch, 12-mesh vibrating screen, and the jig hutch product is fed to No. 8 Wilfley recleaner table. The table concentrate is routed to the drier as a finished product and the table tailing pumped to a 4- by 6-foot, 30-mesh vibrating screen.

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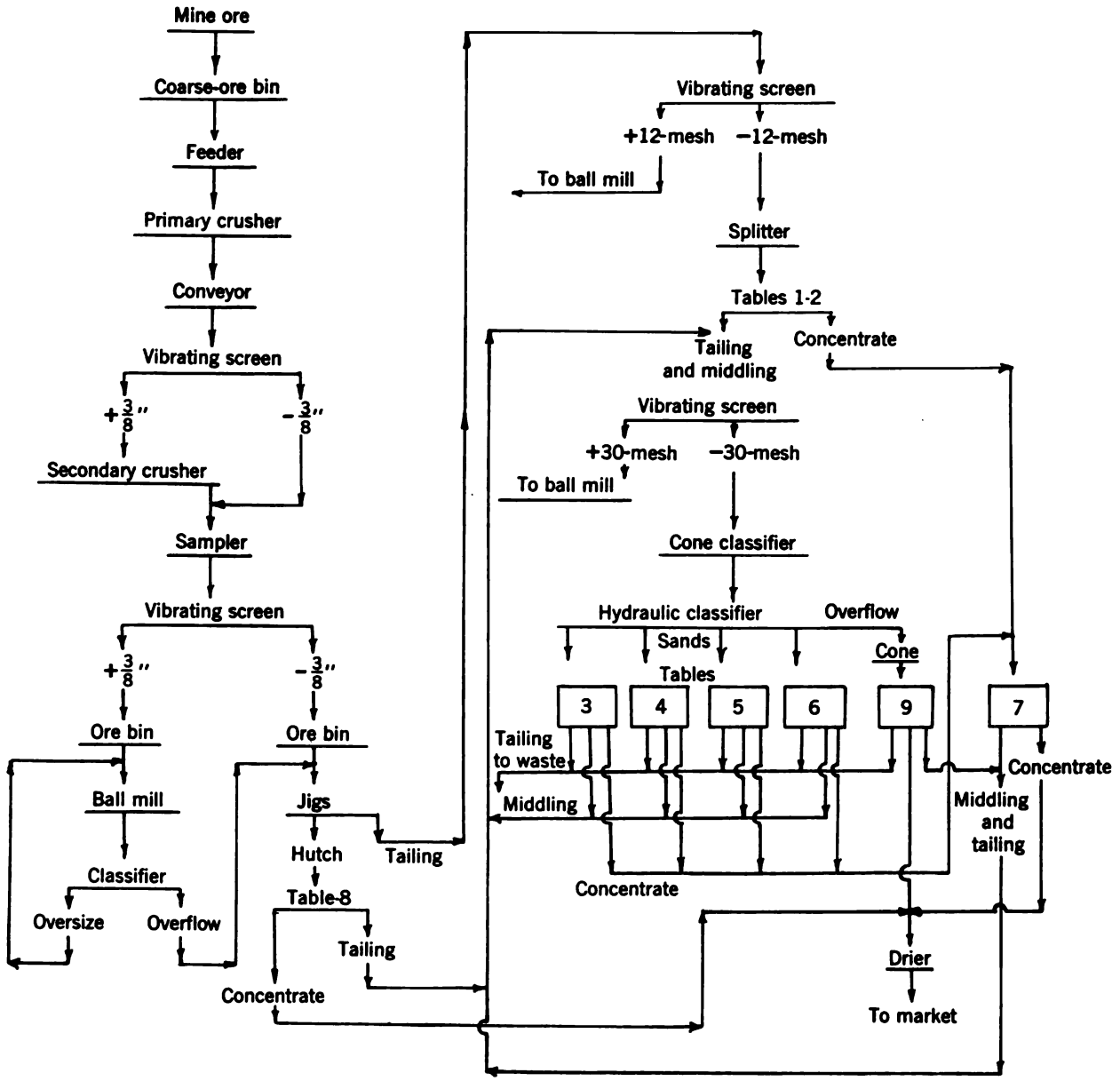

FIGURE 14. - Flowsheet, Minado Corp.

Table Concentration

The jig tailing is sized by a vibrating screen into plus- and minus-12-mesh products. The plus-12-mesh product is returned to the ball mill for regrind and the minus-12-mesh product routed to a splitter that feeds scalping tables 1 and 2. The scalping tables' tailings and middlings are combined and pumped to a splash board and spreader, feeding a 4- by 6-foot, 30-mesh vibrating screen. The plus-30-mesh product returns to the ball mill for regrind, and the minus-30-mesh product is fed to a dewatering cone, which feeds a five-spigot hydraulic classifier. The combined scalping table concentrates are pumped to table 7 for final cleaning. The concentrate from table 7 goes to the drier as a finished product; the middling and tailing are returned to the 30-mesh vibrating screen.

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The four spigot products from the hydraulic classifier are fed separately to scavenger tables 3, 4, 5, and 6, while the fifth spigot product, which constitutes the slime, flows to a 6-foot dewatering cone and then to No. 9 Wilfley recleaner table. The tailings from all five tables are combined, sampled, and pumped to waste; the middlings are returned to the 30-mesh vibrating screen by way of pumps. The combined concentrates from the four scavenger tables are pumped to table 7 for recleaning, and the concentrates from tables 7 and 9 go to the drier as a finished product. The dried concentrates are sacked for shipment to market.

Wah Chang Mining Corp.

Lincoln Division

The Lincoln mill of the Wah Chang Mining Corp. (Lincoln Division) is at Tempiute, approximately 45 miles north of Hiko and 84 miles west of Caliente, Lincoln County, Nev. The ore from the mine 600-foot level is transported in a train of ten 2-ton cars drawn by a diesel locomotive to an open stockpile close to the crushing plant. Ore from the mine 100- and 300-foot levels is truck-hauled to the same stockpile. The general nature of the ore is tactite containing scheelite, calcite, garnet, epidote, apatite, chlorite, molybdenite, sphalerite, pyrite, pyrrhotite, quartz, fluorite, and minor amount of gypsum, oxidized minerals, limonite, and ferrous sulfate.

Milling is conducted in five steps - crushing and grinding, sulfide flotation and gravity concentration, treatment of gravity concentrates, flotation of scheelite-bearing slimes, and lastly acid treatment for scheelite-fluorite separation, because the complex ore contains mixed sulfides, fine-grained scheelite, and the objectionable calcium fluorite. The mill treats 700 to 750 tons of ore per 24 hours. The average mill feed assays 0.37 percent WO_3 and the tailing 0.10 percent WO_3. Recovery approximates 70.0 percent of the contained WO_3 in the ore. Figure 15 is the flowsheet of the milling operations.

Crushing and Grinding

A three-drum, 50-hp., Ingersoll-Rand slusher feeds the ore from the open stockpile onto a 42-inch by 17-foot Stephens-Adams pan feeder that conveys the product to a 42- by 48- inch Traylor jaw crusher. A 30-inch conveyor delivers the crushed product to a 650-ton coarse-ore bin, where a Type 4 Jeffery vibrating feeder feeds a 24-inch conveyor belt that delivers the crushed product to a 3-inch grizzly. The grizzly oversize product is fed to a 12-inch Traylor bulldog gyratory crusher with an opening set at 2-1/2 inches. The grizzly undersize joins the Traylor crusher product on a 20-inch conveyor belt, then is conveyed to a 4- by 10-foot, double-deck, Allis-Chalmers ripple flow screen. The top deck has 2-inch-square openings; the lower deck has 1/2- by 1-1/2-inch Tong-cap openings. The oversize from both decks is fed to a 5-1/2-foot Symons short head cone crusher and crushed to 1/2 inch. The lower deck undersize product and the Symons cone crusher product join on a 20-inch conveyor and are discharged to a 400-ton fine-ore bin. Two Hardinge constant-weight feeders feed the ore from the fine-ore bin separately to two 16-inch conveyor belts that deliver the ore to two 5- by 10-foot Traylor rod mills. Rod mill 1

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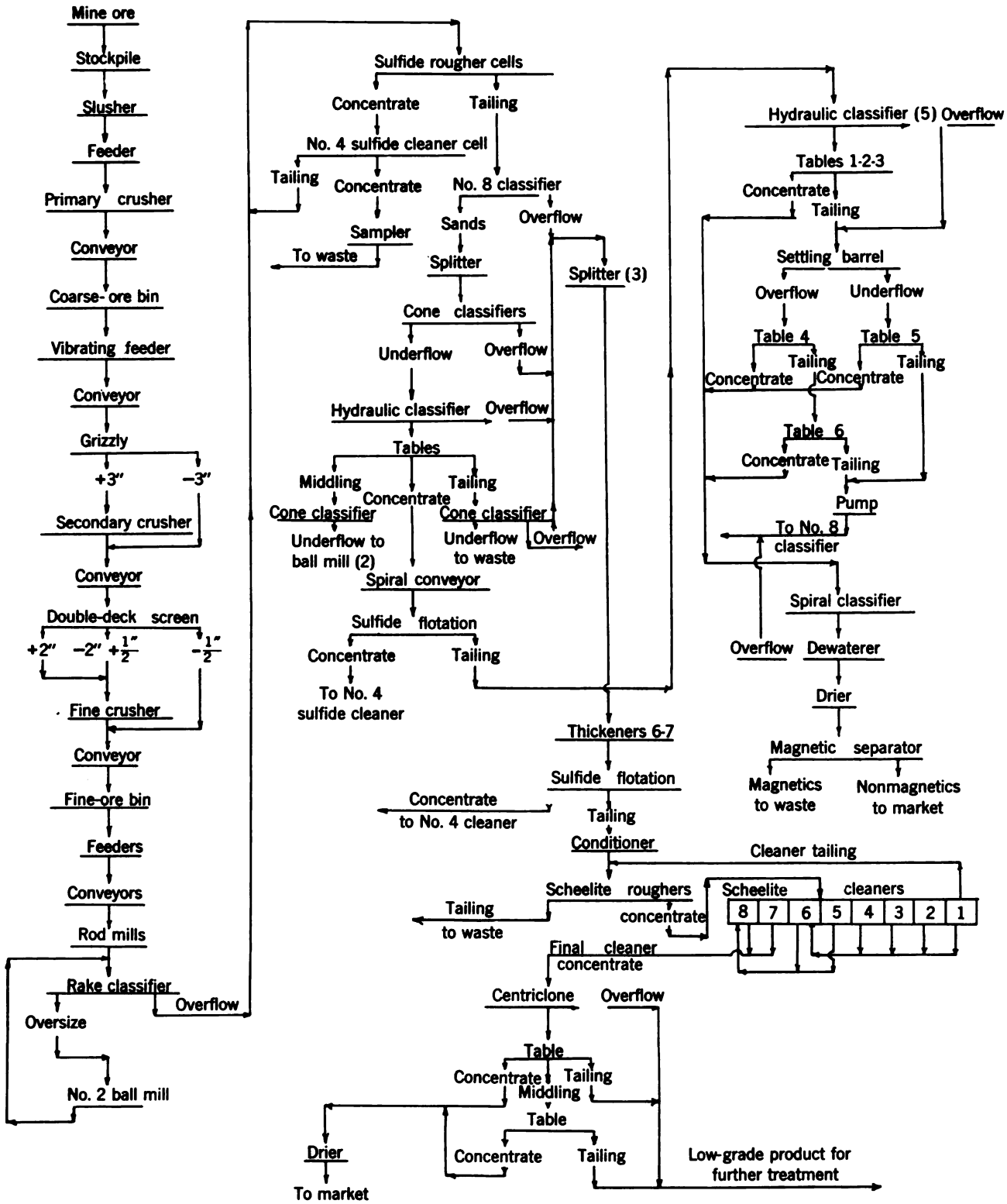

FIGURE 15. - Flowsheet, Wah Chang Mining Corp., Lincoln Division.

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discharges to a 3-inch Wilfley pump. The ground pulp is pumped to a Dorrco 8- by 18-foot 4-inch Type F duplex rake classifier. Rod mill 2 discharges to a launder that flows the pulp by gravity to the same classifier. The sand product from the rake classifier flows to a 7- by 5-foot Allis-Chalmers ball mill, which is in closed circuit with the rake classifier. The classifier overflow is routed to the sulfide flotation and gravity-concentration section.

Sulfide Flotation and Gravity Concentration

The Dorrco rake-classifier overflow is fed to six 62-inch Fagergren rougher cells for sulfide flotation. The sulfide concentrate flows to a 2-1/2-inch Wilfley pump and is pumped to No. 4 sulfide cleaners comprised of two 62-inch Fagergren cells. The cleaner tailing returns to the head of the rougher circuit and the concentrate is sampled and pumped to waste with a 2-inch Hydroseal pump. The rougher tailing flows to a 4-inch Wilfley pump and is pumped to No. 8 classifier that is a 6- by 32-foot Wemco duplex spiral classifier. The classifier sands flow to a 4-inch Wilfley pump and are pumped to a two-way splitter that feeds two 4-foot-diameter, 60°, Apex sand cones. The classifier overflow, which constitutes the slimes, flows to a 3-inch Wilfley pump and is pumped to splitter 3, which distributes the slimes to two Dorrco thickeners (6-7) feeding the slime circuit.

The underflow from the two sand cones flows separately to two St. Joe-type (10-spigot each) Wemco hydraulic classifiers. The 20 spigot products are fed separately to 20 standard, Deister, superduty, shaking tables. Each table produces a concentrate, a middling, and a tailing, and like products of each table are combined. The combined tailings flow to two 3-inch Wilfley sand pumps and are pumped to a 4-foot-diameter, 40°, Apex sand cone. The cone underflow goes to waste. The combined middlings flow through a 3-inch Wilfley pump to another 4-foot-diameter, 40° Apex sand cone. The middling cone underflow is routed to ball mill 2 for regrind. Overflows from all of the cones and the two hydraulic classifiers are combined and flow to splitter 3 for distribution to the two Dorrco thickeners that feed the slime flotation circuit. The combined table concentrates flow to a 6-inch spiral conveyor, which carries them to four 15-inch Galigher sulfide scavenger cells for further flotation of sulfide minerals. The sulfide concentrate is pumped by a 1-1/2-inch Kimball-Krough pump to No. 4 sulfide cleaner circuit.

Gravity-Concentrate-Treatment Section

The tailing from the sulfide scavenger circuit flows to a 2-inch Hydroseal pump and is pumped to a four-spigot, St. Joe-type, hydraulic classifier, designated as No. 5. The four-spigot products are fed to three Deister diagonal tables; the last two-spigot products are combined to feed table 3. Table 3 tailing and the hydraulic classifier overflow are combined; they flow to a 2-inch Wilfley pump and are pumped to a settling barrel. The overflow from the settling barrel is fed to table 4 and the underflow to table 5. The tailing from table 4 is fed to table 6. Tailings from tables 5 and 6 are combined and flow to a 2-inch Wilfley pump for delivery to No. 8 Wemco classifier. The concentrates from all 6 tables are combined and flow to a 9-inch spiral classifier; they then discharge to a 1-inch Wilfley pump, and the concentrate is

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pumped to a 9-foot dewatering classifier. The dewatered product then goes to a 9-inch by 22-foot double-screw infrared drier, and the dried product is fed to a Stearns three-pole magnetic separator. The iron is removed at the first pole, and the garnet at the second and third poles. The nonmagnetic product is sacked for shipment.

Slime-Treatment Section

The combined slimes resulting from the gravity circuit are routed to splitter 3. They are equally distributed to two 12- by 45-foot Dorrco thickeners (6 and 7). The overflow from both thickeners is used as reclaimed water. The underflow from each thickener is pumped by two 4-inch Dorrco triplex diaphragm pumps to four 66-inch Fagergren sulfide scavenging cells. The sulfide concentrate flows to a 2-inch Wilfley pump and is pumped to No. 4 sulfide cleaner circuit for further cleaning. The tailing flows to a 4-inch Wilfley pump and is pumped to a 10-foot Galigher conditioner in preparation for the fatty acid flotation. The conditioned pulp then flows to eight 56-inch, Fagergren, scheelite rougher cells. The rougher tailing goes to waste, and the concentrate flows to eight No. 18 special Denver cleaners, entering the cleaner circuit at cells 5 and 6. The cleaner circuit operates countercurrent for a three-stage cleaning. The cleaner tailing flows to a 2-inch Wilfley pump and is returned to the head of the rougher circuit. The final cleaner concentrate flows to a 2-inch Wilfley pump and is pumped to a 10-inch centriclone. The centriclone underflow is fed to a No. 6 Deister superduty diagonal shaking table, which yields a concentrate, a middling, and a tailing. The table middling is fed to another table which produces a concentrate and a tailing. The tailings from both tables, plus the centriclone overflow, are low-grade products that are further treated by a secret process for the scheelite-fluorite separation. The clean concentrate from both tables is routed to a pan drier, and the dried product is sacked for shipment. Table 22 shows a typical analysis of a monthly composite of the combined tailings, and table 23 the flotation-reagent consumption and places of addition.

TABLE 22. - Typical analysis of a monthly composite of the combined tailings, Wah Chang Mining Corp., Lincoln Division

Mesh	Weight-percent	WO_3 percent	Distribution, percent
35.....	5.8	0.092	4.8
48.....	9.9	.114	10.1
65.....	14.8	.118	15.7
100.....	11.5	.114	11.8
150.....	10.9	.109	10.7
200.....	5.3	.084	4.0
-200 sands.....	27.9	.071	17.8
-200 slimes.....	13.9	.202	25.1
Total.....	100.0	.111	100.0

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TABLE 23. - Flotation-reagent consumption and place of addition, Wah Chang Mining Corp., Lincoln Division

	Reagent	Original feed, pounds per ton	Place of addition
Main sulfide flotation.	Xanthate 301.....	0.80	Mills and staged to cells
	Aerofloat 25.....	.07	Do.
	MIBC.....	.06	Cells
	Copper sulfate.....	.40	Do.
Table sulfide scavenger.	Copper sulfate.....	.04	Cells
	Aerofloat 25.....	.01	Do.
	MIBC.....	.01	Do.
	Xanthate 301.....	.10	Do.
	NaOH.....	.01	Do.
Slime sulfide scavenger.	MIBC.....	.01	Cells
Scheelite flotation.	Quebracho.....	.80	Conditioner
	Soda ash.....	4.0 to 7.0	Do.
	Sodium silicate.....	5.00	Do.
	Sodium cyanide.....	.40	Do.
	Oleic acid.....	.20	Staged to cells
	Emcol 4150 (detergent)	.06	Do.
Scheelite cleaner.	Quebracho.....	.01	Cells
	Sodium silicate.....	.40	Do.
Table conditioning.	Quebracho.....	.02	To centriclone
	Lime.....	.20	Do.

Benton Division

The Benton mine and mill of the Wah Chang Mining Corp., formerly known as the Black Rock mine, is 32 miles north of Bishop, on Highway 6 in the Indian mining district, Mono County, Calif. The ore from the Benton mine is truck-hauled to the mill stockpile. The ore is a tactite containing scheelite, calcite, fluorite, apatite, epidote, garnet, pyrite, pyrrhotite, and minor amounts of chalcopyrite. One section of the mine contains appreciable fluorite. Ore from this section causes difficulties in the flotation circuit; with such ore, it is difficult to achieve more than a 50-percent WO_3 concentrate, even after leaching to remove calcite.

The scheelite is concentrated by flotation, followed by acid leaching of the scheelite product to remove the calcite and apatite. The leached product is then blended with concentrates from the company Lincoln mill to produce one concentrate for shipping. The mill feed ranges from 0.30 to 0.60 percent WO_3, and the average mill recovery is approximately 80 percent. The cleaned flotation concentrate assays 24 to 34 percent WO_3. The mill treats 400 to 500 tons of ore per 24 hours. An accumulation of tailings from the old Tungstar mill is being mixed with pit and underground ores in about the ratio of four tailings, three pit ores, and three underground ores. Some of the mine ore is being truck-hauled to Union Carbide & Nuclear Co. mill at Pine Creek for treatment. Figure 16 shows the flowsheet of the milling operations.

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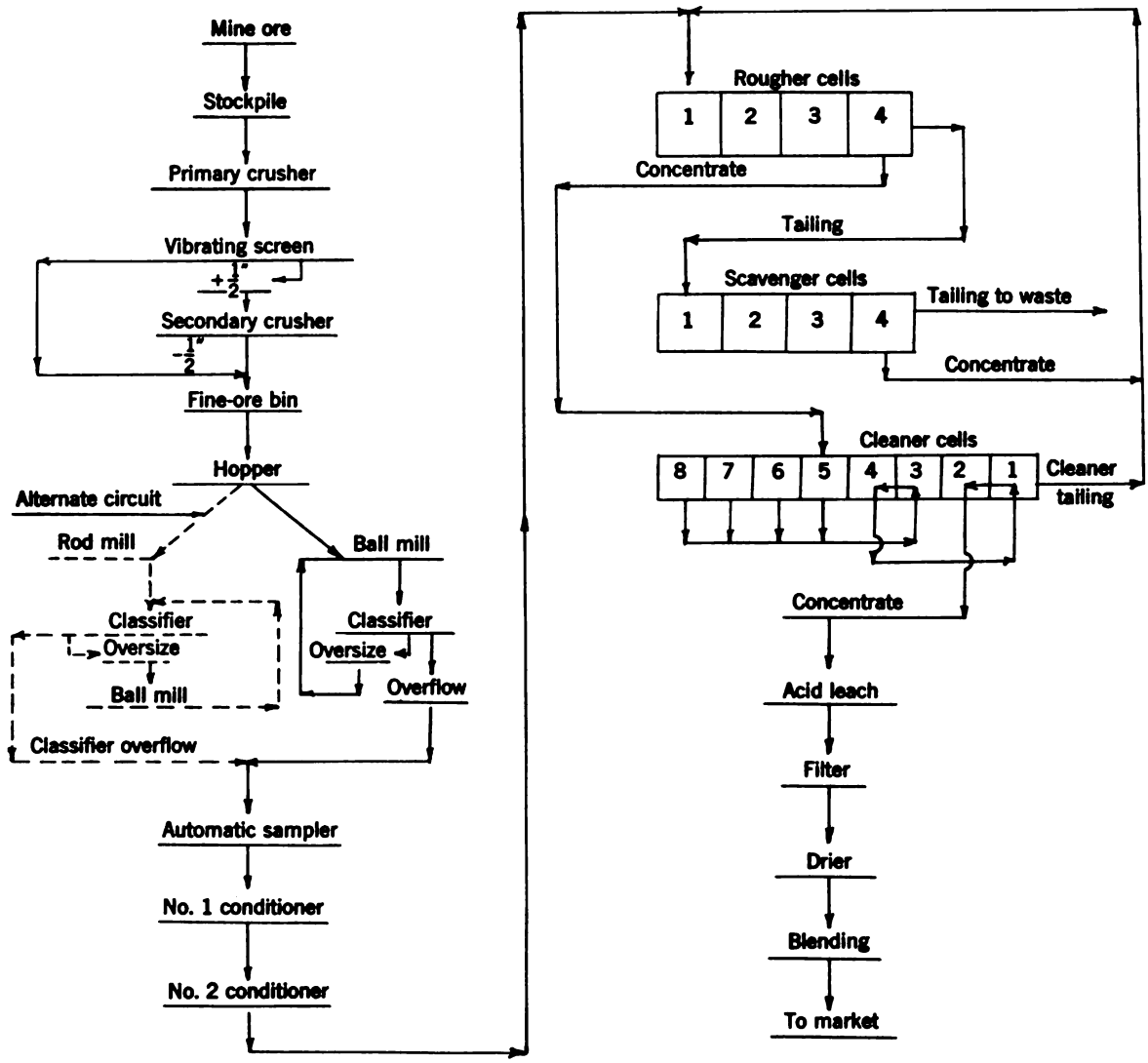

FIGURE 16. - Flowsheet, Wah Chang Mining Corp., Benton Division.

Crushing and Grinding

The ore from the mine is truck-hauled to a stockpile near the primary crushing plant. A skip loader feeds the ore from the stockpile to a 24- by 36-inch Austin Western jaw crusher. The crushed product is conveyed to a 4- by 6-foot Symons vibrating screen having 1/2-inch openings. The screen oversize is fed to a 2-foot, 4-inch Traylor TY crusher set at 5/8 inch. The secondary crusher product joins the screen undersize and is conveyed to a circular fine-ore bin of about 500-ton capacity. A Hardinge constant-weight feeder delivers the ore to a conveyor belt, which discharges into a hopper. From the hopper the ore can be fed to either of two grinding circuits. One grinding circuit consists of a 10- by 3-foot Hardinge overflow-discharge ball mill in closed circuit with an 8-foot Dorr duplex classifier. The other circuit consists of a 4- by 10-foot Hardinge rod mill, which discharges into a 6-foot classifier.

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The classifier oversize is fed to a 6- by 6-foot Allis-Chalmers ball mill, which is in closed circuit with the classifier. The classifier overflow from either grinding circuit contains solids of 95 percent minus-65-mesh and 30 percent plus-200-mesh and is pumped to No. 1 conditioner. The pulp is sampled automatically as it enters the conditioner. The pulp from conditioner 1, which is a 10- by 10-foot Galigher, flows by gravity to a 6- by 6-foot No. 2 Denver conditioner.

Flotation

The conditioned pulp from conditioner 2 flows to four 62-inch Fagergren rougher cells. The tailing from the rougher circuit flows to four 66-inch Fagergren scavenger cells. The concentrate from the scavenger circuit returns to the head of the rougher circuit; the scavenger tails go to waste. The concentrate from the rougher circuit goes to eight Denver cleaners for a three-stage cleaning. The cleaner tails are returned to the rougher circuit. The final cleaner concentrate pumps to three 8- by 10-foot, rubber-lined, conical agitators for the acid treatment. Hydrochloric acid is added to remove the calcite and apatite and the resulting product water-washed by decantation. The leached concentrate is filtered in a Shriver filter press and dried in an infrared-heated double-screw conveyor. The dried product then goes to the blending room where it is mixed with concentrate produced at the company Lincoln mill at Tempiute, Nev., to produce a marketable product.

Table 24 is a typical analysis of the classifier overflow, table 25 shows a typical analysis of the final concentrate, and table 26 shows the flotation-reagent consumption and places of addition.

TABLE 24. - Typical screen analysis of the classifier overflow, Wah Chang Mining Corp., Benton Division

Mesh	Weight-percent	WO_3, percent
+ 65	4.3	0.23
+100	5.9	.08
+150	8.2	.05
+200	10.5	.17
+325	17.7	.41
-325 sands	40.5	.50
-325 slimes	12.9	.58

TABLE 25. - Typical analysis of final concentrate, Wah Chang Mining Corp., Benton Division

Concentrate	Analysis, percent
Tungsten trioxide	60.48
Molybdenum	.84
Phosphorus	.02
Copper	.01
Sulfur	.41

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TABLE 26. - Flotation-reagent consumption and place of addition, Wah Chang Mining Corp., Benton Division

Reagent	Pounds per ton	Place of addition
Soda ash	13.00	Ball mill
Sodium cyanide	1.00	Do.
Sodium silicate	7.50	No. 1 conditioner
Quebracho	.50	Do.
Oleic acid	.25	No. 2 conditioner
Do	.25	Rougher, stage-fed
Frother 77 and blown tall oil (1-1)	.15	Rougher and scavenger
No reagents to cleaner circuit		

Minerva Scheelite Mining Co.

The Minerva Scheelite Mining Co. operates the Chief and Silver Bell mines, 47 miles south of Ely and 15 miles east of Highway 93 on a gravel road in White Pine County, Nev. The ores from the mines are truck-hauled 3 miles north to the small Minerva gravity custom mill, capacity, 1-1/2 tons per hour. The ore is quartz containing scheelite and calcite. The grade of ore ranges from 0.5 to 5.0 percent WO_3 and averages nearly 1.5 percent WO_3. The concentrates produced from different stages of gravity operation are blended to produce a marketable product that contains 65 percent or more WO_3. The recovery attained at the mill is between 80 and 85 percent of the total tungsten in the mill feed. Figure 17 is the flowsheet of the milling operations.

Crushing and Grinding

Ore from the mine is truck-hauled to the mill and dumped into a 25-ton coarse-ore bin, from which a conveyor feeds the ore to an 8- by 15-inch Joshua-Hendy jaw crusher for reduction to 1-1/4 inches. The crushed product is discharged to a bucket elevator for delivery to the fine-ore bin. The fine-ore-bin product is fed to a double-deck vibrating screen with 1- and 1/4-inch screen cloths. The 1-inch oversize product is stored in a bin for future hand sorting of scheelite-bearing material. The minus-1-inch, plus-1/4-inch product is fed to a 9-1/2-inch Kennedy Van Saum gyratory crusher set at 1/4 inch. The crushed product is fed to a 2- by 6-foot Marcy rod mill. The screen minus-1/4-inch product joins the rod mill discharge at the elevator and is discharged to a 16-mesh trommel screen. The trommel screen oversize returns to the rod mill for regrind, and the minus-16-mesh product is routed to the rougher table.

Table Concentration

The minus-16-mesh product is fed to a 5- by 15-foot Wilfley table, which produces a concentrate, a middling, and a tailing. The tailing flows to waste, the concentrate is 70-percent WO_3 finished product, and the middling product is pumped to a 40-mesh trommel screen.

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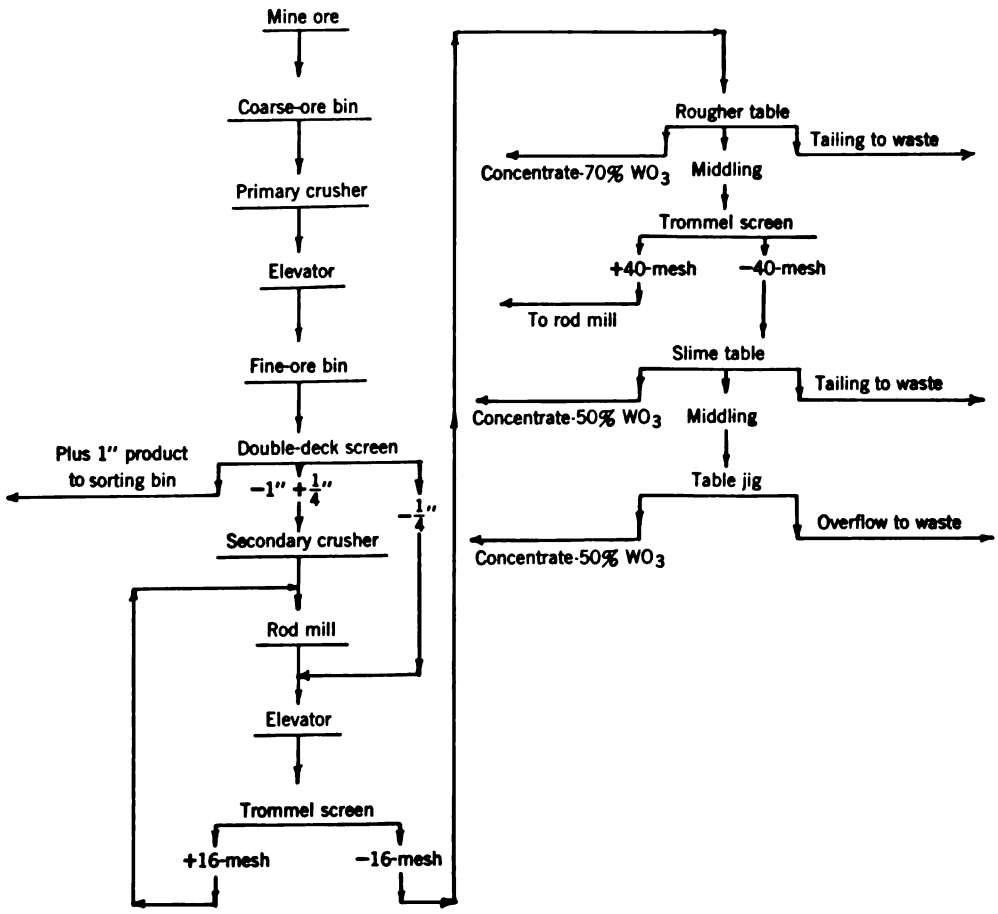

FIGURE 17. - Flowsheet, Minerva Mill, Minerva Scheelite Mining Co.

The plus-40-mesh screen product is returned to the rod mill for regrind and the minus-40-mesh product is fed to a Wilfley slime table, which also produces a concentrate, a middling, and a tailing. The slime-table tailing flows to waste, the concentrate constitutes a 50-percent WO_3 finished product, and the middling is fed to a table jig.

The table jig is an invention of Robert Stopper, owner and operator of the Minerva custom mill, and Martin Lensch, mill foreman. It is a rectangular pan approximately 4 feet long, 12 inches wide, and 1 inch deep, mounted level and equipped with a head motion similar to the head motion for standard concentrating tables. The shaking speed or strokes per minute applied to the table jig depend on the particle size of the material being treated. The shaking speed is extremely critical for effective concentration. The slime-table middling fed to the table jig averages 1.38 percent WO_3. The screen analysis of the slime-table middling product shows that 6.0 percent of the total weight of the material is in the minus-200 plus-325-mesh fraction and assays 0.15 percent WO_3. Twenty-five percent of the total weight of the material is in the minus-325-mesh fraction and assays 0.38 percent WO_3. With proper operation of the table jig, it is possible to recover the fine tungsten

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in a concentrate assaying 50 percent WO_3. Approximately 10 percent of the total tungsten in the mill feed is contained in the slime table middling which is recovered in the table jig. The table-jig concentrate is permitted to settle in the bottom of the table jig while the tailings overflow to waste. When enough concentrated material is accumulated in the bottom of the table jig, it is scraped out by hand, dried, and blended with the higher grade table concentrate to produce a final concentrate product that assays 65.0 or more percent WO_3.

Metallurgical Development Co.

The mine and mill of the Metallurgical Development Co. are 12 miles east of Gardnerville in the Pine Nut district, Douglas County, Nev. Approximately 50 percent of the ore milled is company ore; the remainder is custom ore from the Alpine mine in California and small producers in the vicinity of Rawhide, Nev. The ores treated are scheelite, with the associated minerals - calcite, garnet, apatite, epidote, pyrite, marcasite, molybdenite, magnetite, powellite, and quartz in silicified limestone, mainly tactite.

The tungsten minerals are concentrated by jigging, tabling, and flotation, followed by roasting and magnetic separation. The mill treats 75 to 100 tons of ore per 24 hours, producing a final concentrate containing more than 60 percent WO_3. The mill feed averages 0.6 percent WO_3, and the tailing 0.04 percent WO_3, giving an overall recovery of more than 90 percent. Figure 18 is the flowsheet of the milling operations.

Crushing, Grinding, and Gravity Separation

The mine and custom ores are truck-hauled to the mill and dumped into a hopper-type bin that feeds a 10- by 36-inch Cedar Rapid jaw crusher. The crushed ore is elevated and fed into a 6-inch Traylor gyratory crusher set at 1 inch. The gyratory crusher product is elevated to the fine-ore bin, and from there a belt feeder delivers it to a wet elevator, which discharges it to a trommel screen with 1/4-inch openings. Oversize goes to a 3- by 8-foot Allis-Chalmers rod mill and undersize to an 18-inch Bendelari jig. The jig tailings go to a 20-mesh Tyler screen. Oversize from the Tyler screen is returned to the rod mill for regrind. The rod mill is in closed circuit with the jig. The hutch product from the jig goes to a 6- by 12-foot Daniels diagonal table. The table tailing is returned to the rod mill for regrind. The table concentrate goes to a dewatering drag, roaster, and magnetic separation.

The undersize from the 20-mesh Tyler screen goes to a cone classifier that classifies the product into coarse and fine sand products. The coarse and fine sands are fed separately to two 6- by 12-foot diagonal tables, each table producing a concentrate, a middling, and a tailing. Like products from both tables are combined. Middlings are pumped to the rod mill for regrind, tailings and the cone slime overflow to the flotation thickener, and the concentrates to the dewatering drag.

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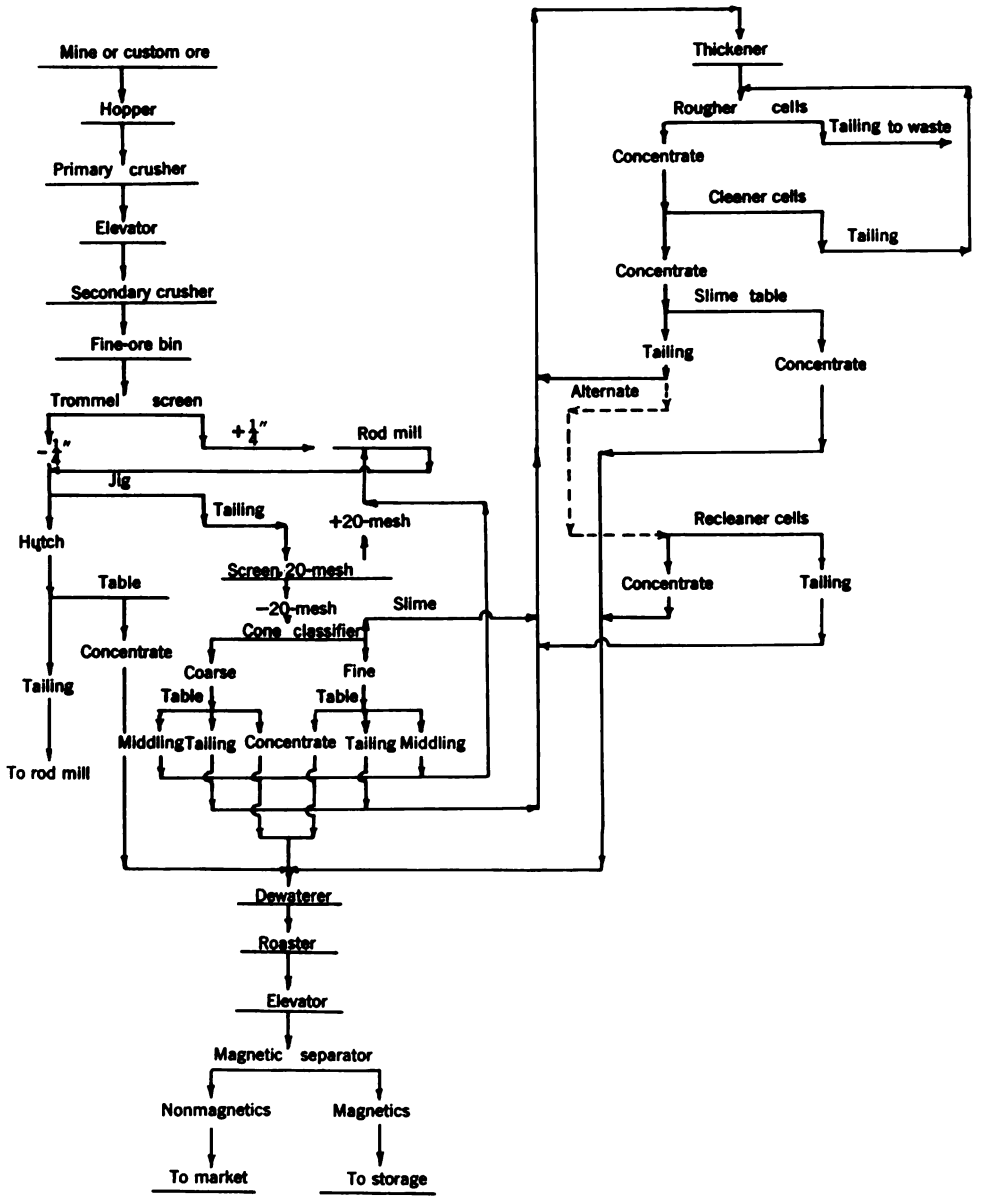

FIGURE 18. - Flowsheet, Metallurgical Development Co.

Flotation

The combined table tailings and cone slimes are thickened and the pulp fed to a rougher circuit of five 36-inch Denver cells; the feed entering the circuit at cell 3. The rougher tailing goes to waste and the rougher concentrate to a 36-inch Denver cleaner. The cleaner tailing is recirculated to the head of the rougher circuit and the concentrate to a slime table. The slime table concentrate goes to the dewatering drag, and the tailing is returned to the thickener. If the slime-table product contains sulfides or is not high grade enough to meet specifications, an alternate circuit is used either to float sulfides or reclean the concentrate. Reagents used are as follows:

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One part of tall oil is mixed with two parts of oleic acid and the mixture fed at the rate of 0.4 pound per ton of ore. Soda ash is added to maintain a pH between 9.5 and 10.0 at the rate of 8 pounds per ton. Other reagents stage-fed, in pound per ton, are: Sodium silicate, 0.01; D-40, 0.005; ferric sulfate, 0.1; methyl amyl alcohol frother, 0.1.

Roasting and Magnetic Separation

The concentrate from the dewatering drag is fed to the roaster without prior drying. The roaster, locally designed, consists of a stainless steel tube 12 inches in diameter and 17 feet in length, chain-driven at 17 r.p.m. and oil-fired. The roasted product passes to a steel bucket elevator that feeds a four-pole Dings Wetherill-type magnetic separator. The garnet and iron sulfides containing approximately 0.6 percent WO_3 are removed as magnetic products and stored for future treatment to recover additional tungsten. The nonmagnetic product is sacked in 100-pound bags for market. Table 27 is a typical analysis of the final tungsten concentrate.

TABLE 27. - Typical analysis of final concentrate, Metallurgical Development Co.

Concentrates	Analysis, percent	Concentrates	Analysis, percent
Tungsten trioxide.....	62.89	Molybdenum.....	0.17
Tin.....	Trace	Phosphorus.....	.11
Copper.....	.02	Manganese.....	.09
Arsenic.....	None	Lead.....	.08
Antimony.....	None	Zinc.....	.02
Bismuth.....	.01		

Red Hill Custom Mill

The Red Hill custom mill is 4 miles west of Bishop, Inyo County, Calif. The custom ores treated are from various sources and largely from contact metamorphic deposits that contain scheelite, garnet, epidote, calcite, pyrite, pyrrhotite, chalcopyrite, and quartz. The scheelite is recovered by gravity methods, followed by roasting and magnetic separation of the gravity concentrate to produce a marketable product of over 60 percent WO_3. The grade of the custom ore varies. The average is about 0.65 percent WO_3. The mill treats 100 tons of ore per 24 hours at an average recovery of 75 percent. Figure 19 is the flowsheet of the milling operations.

General Milling Practice

Custom ores from various sources are truck-hauled to the mill, passed through an 8-inch grizzly into an 80-ton coarse-ore bin, then fed to a 10- by 20-inch Pacific jaw crusher. The crushed product is elevated to a 2- by 4-foot Denver Dillon screen having 1/2-inch openings. The screen undersize is conveyed to the fine-ore bin. The oversize is fed to a 24-inch Allis-Chalmers gyrocrusher in closed circuit with the Denver Dillon screen.

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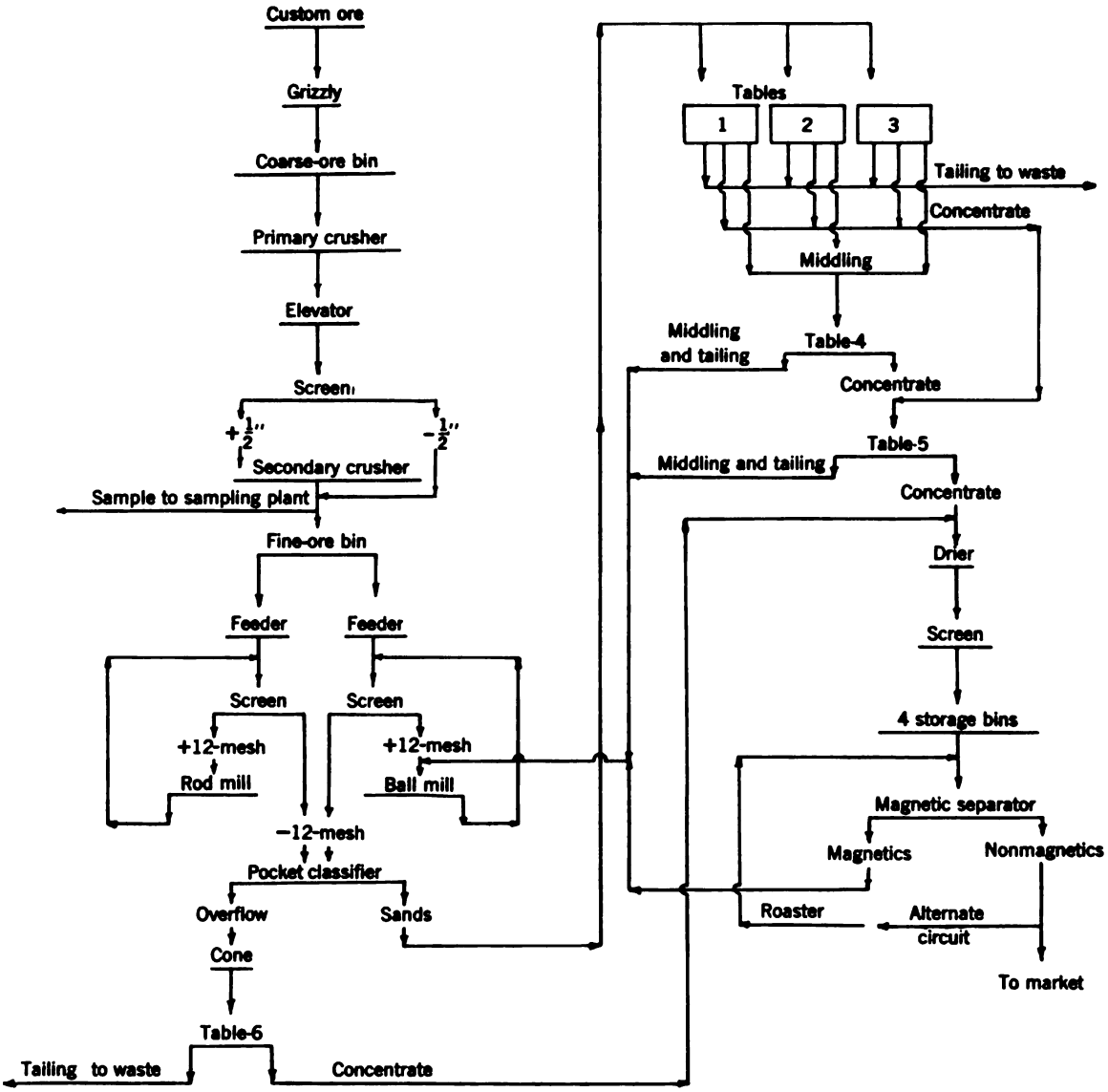

FIGURE 19. - Flowsheet, Red Hill Custom Mill.

An automatic sampling system with triple Snyder cutters samples the minus-1/2-inch stream to the fine-ore bin. The cut portion of the first Snyder goes to a tumbling barrel, then to the second Snyder cutter. The cut portion from the second sampler passes through a 4- by 6-inch jaw crusher for reduction to 3/16 inch. The crushed product is then cut by the third Snyder for the final sample. The rejects from all sample cutters go to the fine-ore bin by way of the elevator.

The fine-ore bin is equipped with dual feeders, one feeding the ball-mill circuit and the other the rod-mill circuit. The feeders deliver the ore to 3- by 6-foot Cottrell 12-mesh screens. The oversize from the screens goes to the 2- by 3-foot Straub ball mill or to the alternate 3- by 6-foot Marcy rod mill,

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both of which are in closed circuit with the screens. The screen undersize goes to a pocket classifier. The classified sands are distributed to three 16-foot Straub rougher tables, each table producing a concentrate, a middling, and a tailing. Like products of all three tables are combined with the tailings going to waste, and the middlings are fed to Wilfley table 4. Middlings from table 4 are returned to be reground and the concentrates from all four tables are fed to table 5 for recleaning. Tailing and middling from table 5 are returned to be reground; the concentrate goes to a drier.

The slime overflow from the pocket classifier is fed to a settling cone, then to table 6, which produces a concentrate and a tailing. The concentrate goes to the drier and the tailing to waste.

The dried product is screened into minus-40-, minus-30-, minus-12-, and plus-12-mesh products. Each product is stored separately in a four-compartment bin. Each sized product is treated separately on a three-pole Stolle magnetic separator. The magnetic product is returned to regrind. If the nonmagnetic product contains sulfides, it is roasted and retreated on the magnetic separator.

Ajax Tungsten Corp.

The Benware mill, 6 miles south of Bishop, Inyo County, Calif., is owned by the Molybdenum Corporation of America. During 1955 and 1956, the milling facilities were leased to the Ajax Tungsten Corp. for the treatment of test lots of tungsten ores from its Durham mine in the Darwin district. The magnetic separation section of the mill is also utilized by the Molybdenum Corporation of America, largely for custom work in cleaning and upgrading concentrates and other products from various mills throughout the United States, Canada, and Mexico.

The ore from the Durham mine is scheelite in tactite containing garnet, epidote, pyrite, calcite, and quartz. The scheelite is concentrated by jigging and tabling, followed by roasting and magnetic separation of the scheelite concentrate. The average mill feed assays 0.70 percent WO_3 and the tailings 0.14 percent WO_3. The mill treats 75 tons of ore per 24 hours.

Crushing and Grinding

Test lots of ore from the Durham mine are truck-hauled to the Benware-mill crushing plant and dumped on a 10-inch-opening grizzly. The minus-10-inch product is fed to a 10- by 24-inch Straub Kue-Ken jaw crusher and reduced to 1 inch. The crushed product is elevated to a conveyor belt that discharges to the fine-ore bin. The ore from the fine-ore bin is fed to another elevator that discharges onto a 2- by 4-foot Denver Dillon screen with 1/4-mesh screen openings. The screen oversize is fed to a 4- by 5-foot, Straub, peripheral discharge ball mill and the undersize to a rougher jig.

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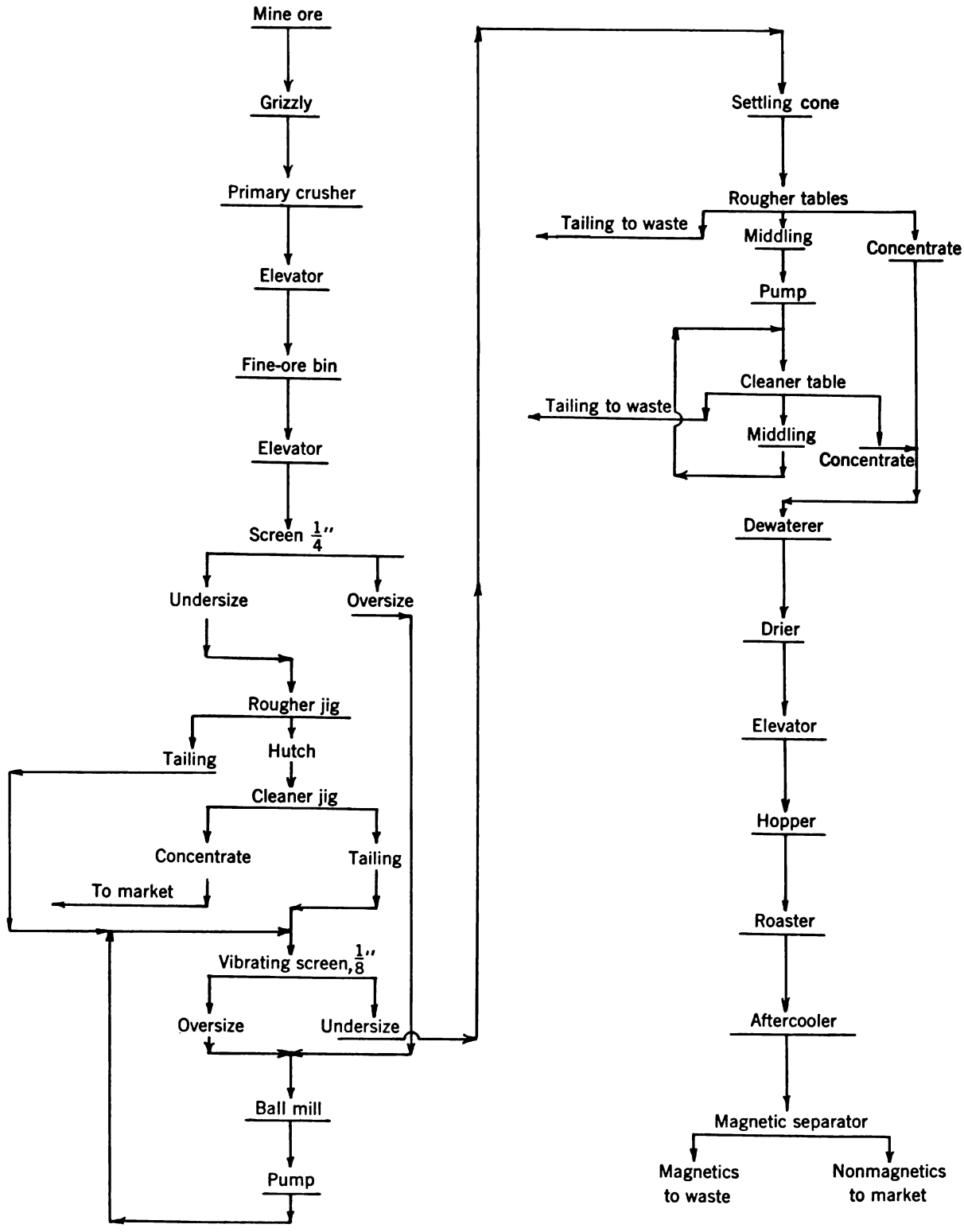

FIGURE 20. - Flowsheet, Benware Mill, Ajax Tungsten Corp.

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Jigging, Tabling, Roasting, and Magnetic Separation

The minus-1/4-mesh Denver Dillon screen undersize is fed to a two-compartment, 14- by 18-inch, Denver duplex jig. The jig bed is equipped with an 8-mesh screen cloth. The minus-8-mesh jig hutch product is fed to an 8- by 12-inch Denver simplex jig for cleaning. The cleaned concentrate is a finished product. Tailings from the rougher and cleaner jigs to to a 3- by 6-foot Straub vibrating screen having 1/8 inch screen openings. The Straub-screen oversize goes to the ball mill, which is in closed circuit with the screen. The screen undersize flows to a settling cone and the unclassified cone feed is distributed to two 15-foot Deister Plato and two modified Dunham-type tables, each table producing a concentrate, a middling, and a tailing. The tailings go to waste, and the middling is pumped to a cleaner table; the cleaner-table tailing goes to waste, and the middling is recirculated to the cleaner table. The concentrates from all five tables are combined, dewatered, and dried in a butane-gas-fired rotary kiln. The dried concentrate is elevated to a hopper feeding a 10-inch by 14-foot, externally butane gas-fired, revolving roaster, thence proceeds to a tubular aftercooler, then to a three-pole Dings crossbelt magnetic separator. The magnetic product is discarded to waste and the nonmagnetic product sacked for shipment.

Union Carbide Nuclear Co.

The mine, mill, and chemical plant of the Union Carbide Nuclear Co. (formerly the U. S. Vanadium Co.), a division of the Union Carbide Corp., are approximately 21 miles northwest of Bishop in the Pine Creek district, Inyo County, Calif.

Approximately one-third of the ore treated is custom ore from various sources, principally California and Nevada. The ore from the company-owned mine usually occurs as tungsten-bearing tactite. Among the identifiable minerals are scheelite, calcite, epidote, fluorite, molybdenite, powellite, garnet, chalcopyrite, bornite, and quartz. The copper and molybdenum minerals occurring in the company ore are recovered as byproducts in the sulfide flotation circuit. The scheelite is concentrated by flotation followed by chemical treatment of the low-grade flotation and finishing-table products.

In addition to the treatment of custom ores, the company purchases low-grade concentrates, middlings, magnetic-separator products, and other tungsten-bearing byproducts from various mills throughout Inyo County for chemical treatment.

The mill treats 1,150 to 1,200 tons of ore per 24 hours, producing a scheelite product that averages 70 percent WO_3. Figure 21 is the flowsheet of the milling and chemical plant operations.

Crushing and Grinding

The company ore is derived from open-pit and underground operations. Primary crushing is performed at the mine. The crushed product is transported to the mill by a mile-long aerial tramway. An apron feeder delivers the ore

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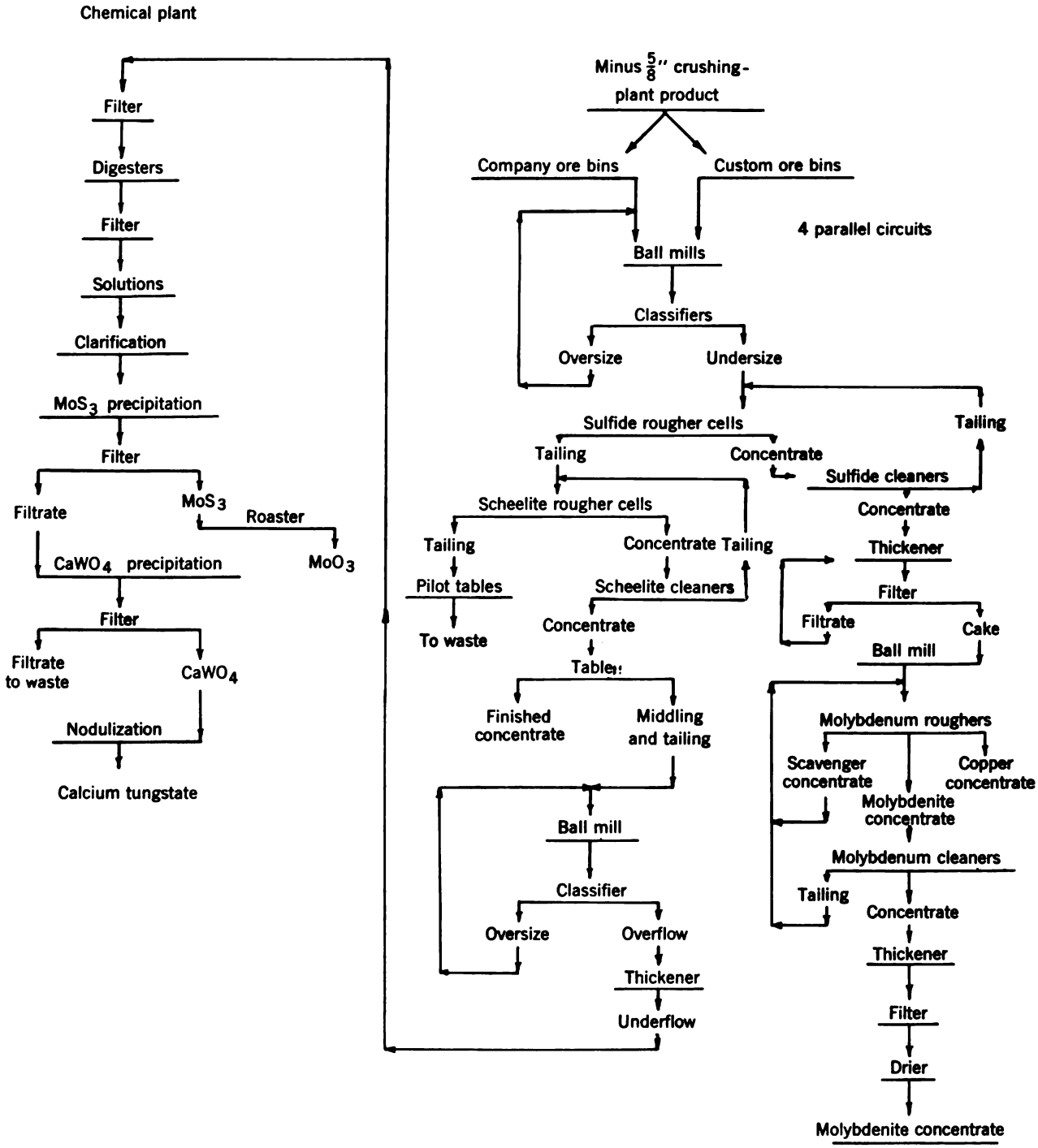

FIGURE 21. - Flowsheet, Union Carbide Nuclear Co.

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from the mill bin to a 32-inch, Symons, rod-deck vibrating screen with 5/8-inch openings. The screen oversize is fed to a 5-1/2-foot, short-head, Symons cone crusher. The crushed product and the screen undersize are combined, passed over a weightometer to the sampling plant, and discharged in two 1,000-ton-each mill surge bins or to an open stockpile.

Custom ores are delivered to the mill by haulage trucks. They pass over a platform scale and are dumped into individual shipper's stockpiles. The ore is milled when approximately 200 tons of ore has been accumulated in the stockpile. The stockpile ore is loaded into haulage trucks by a power shovel and delivered to a jaw crusher for the primary crushing, thence to the same Symons vibrating screen, short-head Symons cone crusher, and sampling plant used for the company ore. The crushed product is stored in two 1,000-ton-capacity custom ore bins for milling.

The grinding and flotation circuits consist of four parallel units. Two units are used for company ore and the remaining two for custom ore. The ore is fed to 6- by 5-foot, Marcy, low-discharge ball mills in closed circuit with 54-inch Akins classifiers and ground to 75 percent minus-100-mesh.

Flotation

The classifier overflow from each grinding unit, which discharges at approximately 55-percent solids, goes to the sulfide roughers, which consist of ten 32- by 32-inch MS cells. Eight cells are used for the sulfide flotation and the last two for conditioning the sulfide flotation tailing before the scheelite flotation. The sulfide flotation concentrate goes to the sulfide cleaners, which consist of five 32- by 32-inch MS cells for a two-stage cleaning. The cleaner tailings are returned to the sulfide rougher circuit. The cleaner concentrate goes to a thickener where it is agitated with steam to break down the contained reagents. The thickener underflow is then filtered and the filter cake ground in a 3-foot ball mill at 70 percent solids. The ground pulp then goes to the molybdenum rougher circuit, which consists of two banks of six cells each of 24- by 24-inch MS cells. The first bank produces a primary concentrate and the second a scavenger concentrate. The copper is depressed with sodium cyanide. The rougher tailing therefore becomes the copper concentrate, which is filtered, dried, and shipped to a copper smelter. The scavenger concentrate is returned to the head of the molybdenum rougher circuit, where the molybdenum concentrate is given six stages of cleaning in a bank of sixteen 24- by 24-inch MS cleaners. The final cleaner molybdenum concentrate goes to a thickener and is filtered. The filter cake is dried in an Edwards-type roaster. The product is marketed as MoS_2. The cleaner tailing is returned to the scavenger circuit. In the scheelite rougher circuit the scheelite is floated in a bank of ten 32- by 32-inch MS cells. The tailings from the circuit go to four Deister diagonal pilot tables to recover any coarse, unfloated scheelite before disposal of tailing. The rougher concentrate is given a two-stage cleaning in five 32- by 32-inch MS cells. The cleaner tailing is returned to the scheelite rougher circuit. The concentrate is then fed to a Deister diagonal table to recover coarse scheelite from the flotation concentrate. The table tailing and middling are reground in a 3-foot ball mill in closed circuit with a Wemco classifier. This finely ground product is pumped to a thickener for the chemical treatment.

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Chemical Treatment Plant

The chemical treatment process was developed by the U. S. Vanadium Co. and is described in U. S. Patent 2,339,888, owned by Union Carbide Corp. A brief description of a similar process is contained in the publication, Tungsten, by K. C. Li and C. Y. Wang, American Chemical Society Monograph Series 94. Basically, the process is a batch operation and is briefly described as follows:

> The pulp in the thickener from the reground table tailings and middlings is filtered and fed to patented pressure digesters, U. S. patent number 2,521,896, owned by the Union Carbide and Carbon Corp., under a closely controlled density. Chemicals and steam are added to the digesters. When digestion is completed, the pulp is filtered and the solution clarified. To the clarified solution is added a soluble sulfide to precipitate the molybdenum as MoS_3. The precipitate is filtered in Sperry pressure filters equipped with wooden frames, and the precipitate dried and given an oxidizing roast to convert the MoS_3 to MoO_3. To the resulting filtrate from the Sperry filter is added calcium chloride to precipitate the tungsten as $CaWO_4$. The precipitated $CaWO_4$ is filtered in Sperry pressure filters equipped with steel frames and the resulting product nodulized in a rotary kiln. The nodulized product is approximately 45 percent plus-4-mesh.

Table 28 gives the reagent consumption, table 29 the screen analysis of the primary classifier overflow, and table 30 a typical analysis of the finished tungsten product.

TABLE 28. - Flotation-reagent consumption and place of addition, Union Carbide Nuclear Co.

Reagent	Pounds per ton	Place of addition
Soda ash	2.0-3.0	Ball mill
Xanthate 301	.05-0.1	Do.
Pine oil	.05-0.1	Sulfide rougher
Sodium silicate	1.0-1.5	Do.
Reagent 710 (American Cyanamid)	1.0-1.5	Stage-fed scheelite rougher
Copper-molybdenum separation: Sodium cyanide, kerosine, and heptonal frother.		

TABLE 29. - Screen analysis of primary classifier overflow, Union Carbide Nuclear Co.

Mesh	Weight-percent	Analysis, percent			Distribution, percent		
		WO_3	Mo	Cu	WO_3	Mo	Cu
+ 48	6.4	0.078	0.29	0.14	1.0	5.9	3.3
+ 65	8.0	.074	.27	.16	1.2	6.8	4.7
+100	11.1	.104	.24	.16	2.3	8.4	6.5
+150	11.9	.150	.25	.18	3.5	9.4	7.9
+200	13.7	.296	.28	.22	8.0	12.1	11.0
+325	12.2	.504	.25	.25	12.1	10.0	11.3
-325	36.7	1.000	.41	.41	71.9	47.4	55.3
Total	100.0	.510	.32	.27	100.0	100.0	100.0

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TABLE 30. - Typical analysis of scheelite concentrate, Union Carbide Nuclear Co.

Concentrate	Analysis, percent	Concentrate	Analysis, percent
Tungsten trioxide..	70.000	Tin.....	Trace
Copper.....	Trace	Antimony.....	0.015
Phosphorus.....	.025	Manganese.....	Trace
Arsenic.....	.025	Sulfur.....	.300
Molybdenum.....	.060	Loss on ignition..	1.000

El Diablo Mining Co.

The El Diablo custom mill is 4 miles west of Bishop, Calif. The ores treated are largely tactite, containing scheelite, epidote, apatite, calcite, garnet, pyrite, quartz, and minor amounts of copper minerals. The scheelite is concentrated by tabling, followed by roasting and magnetic separation of the table concentrates. The grade of the ores treated varies considerably and may range from 0.35 to 5.0 percent WO_3. The mill treats 50 tons of ore per 24 hours. Figure 22 is the flowsheet of the milling operations.

General Milling Practices

Custom ores are truck-hauled to the crushing-plant coarse-ore bin. They are fed to a 10- by 24-inch Pacific jaw crusher and the crushed product elevated to the fine-ore bin, which is fed to an 18-inch Symons disk crusher for the secondary crushing. The secondary product is then fed to a Symons 12-mesh vibrating screen. The screen oversize is fed to a ball mill, and the screen undersize goes to a bucket elevator, which discharges to a 3/16-inch trommel screen. The plus-3/16-inch product goes to the ball mill, which is in closed circuit with a trommel screen. The minus-3/16-inch product is fed to another trommel screen having 3/64-inch openings. The plus-3/64-inch product goes to table 1 for concentrating, and the minus-3/64-inch product goes to a two-spigot hydraulic classifier. The two spigot products are fed separately to tables 2 and 3. The hydraulic classifier overflow is fed to another hydraulic classifier, and the sands from this classifier are fed to table 4. The overflow goes to a 6-foot cone and the contained solids to table 5. The cone overflow goes to waste. All the tables are of the 12-foot Dunham Economy type. The tailings from all the tables are discarded to waste. The combined middlings go to regrind and the combined concentrates to a drier. If the dried concentrate contains sulfides, the concentrate is roasted in preparation for magnetic separation. A three-pole, 18-inch Stolle magnetic separator is used to make a 60-percent or more WO_3 product. The magnetic rejects are either returned to the shipper or discarded, depending on the shipper's wishes.

Round Valley Tungsten Co.

The Round Valley Tungsten Co. mill is 4 miles southwest of Bishop, Inyo County, Calif. The ore from the Round Valley mine is truck-hauled to the mill. The ore is tactite containing scheelite, epidote, garnet, zinc and copper minerals, pyrite, and quartz. Following flotation to remove sulfides, the scheelite is concentrated by tabling, roasting, and magnetic separation. The

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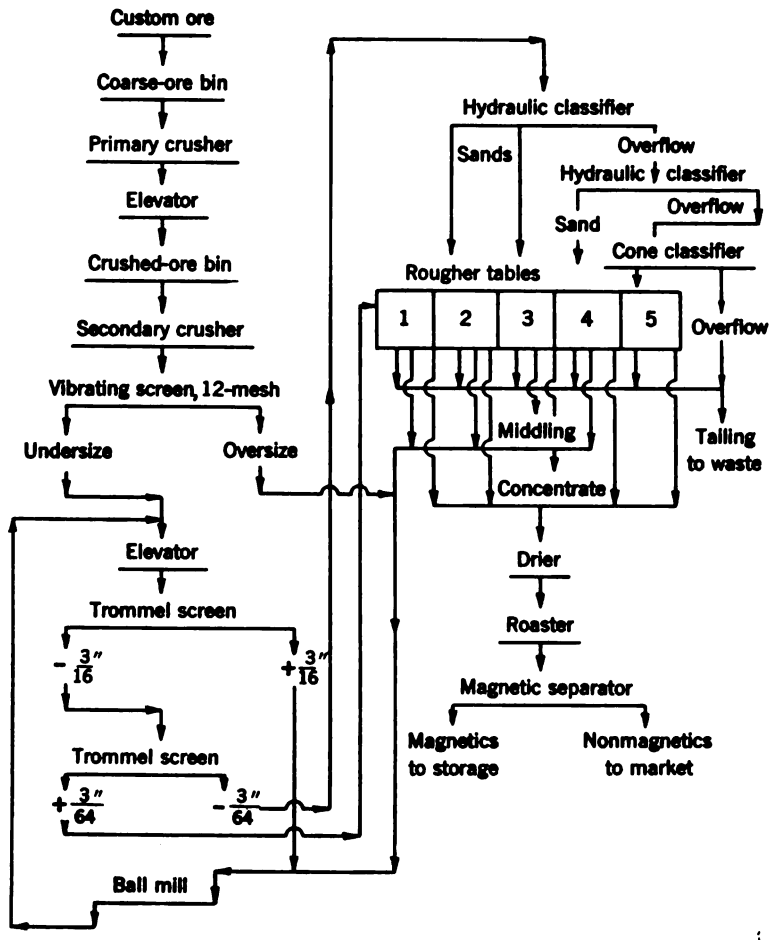

FIGURE 22. - Flowsheet, El Diablo Mining Co.

mill treats 50 tons of ore per 24 hours. The average mill feed assays 0.5 percent WO_3, tailing 0.15 percent WO_3. A marketable concentrate at approximately 70-percent recovery is produced. Figure 23 is the flowsheet of the operation.

General Milling Practice

The ore from the mine is truck-hauled to the mill and dumped into the coarse-bin for delivery to a 3/4-inch stationary screen and a 9- by 15-inch Wheeler jaw crusher for the primary crushing. The crushed product and screen undersize are elevated to a trommel screen with 3/4-inch openings. The plus-3/4-inch product is gravity-fed to a secondary crusher; the secondary crusher product joins the primary crusher product and is elevated to the trommel screen. The minus-3/4-inch product is fed to the fine-ore bin, from which it is conveyed to a 16-mesh vibrating screen. The minus-16-mesh product goes to two conditioners in series for sulfide flotation. The plus-16-mesh product goes to a 3- by 4-foot Straub peripheral-discharge ball mill, which discharges into an elevator in closed circuit with the vibrating screen. Concentrate from the sulfide flotation that contains pyrite, copper and zinc minerals, and small amounts of gold and silver, is stored for future treatment. Tailing from the sulfide float goes to a four-spigot hydraulic classifier whose first three-spigot products constitute the sands that are fed separately to one Wilfley and two Deister tables. The fourth spigot product, which constitutes the slimes, is fed to two Dunham Economy slime tables. Tailings from all five tables go to waste. Middlings from the first three tables go to Wilfley table 6. Middlings and slimes from the two slime tables are stored for shipment. Tailing from table 6 is returned to the ball mill for regrind, and the concentrate from all the tables go to a drier, and, if necessary, to roasting before magnetic separation. The roasting is conducted in a rotary kiln externally heated with butane gas and automatically controlled at any desired temperature. After cooling the roasted product goes to a vibrating screen and is separated into

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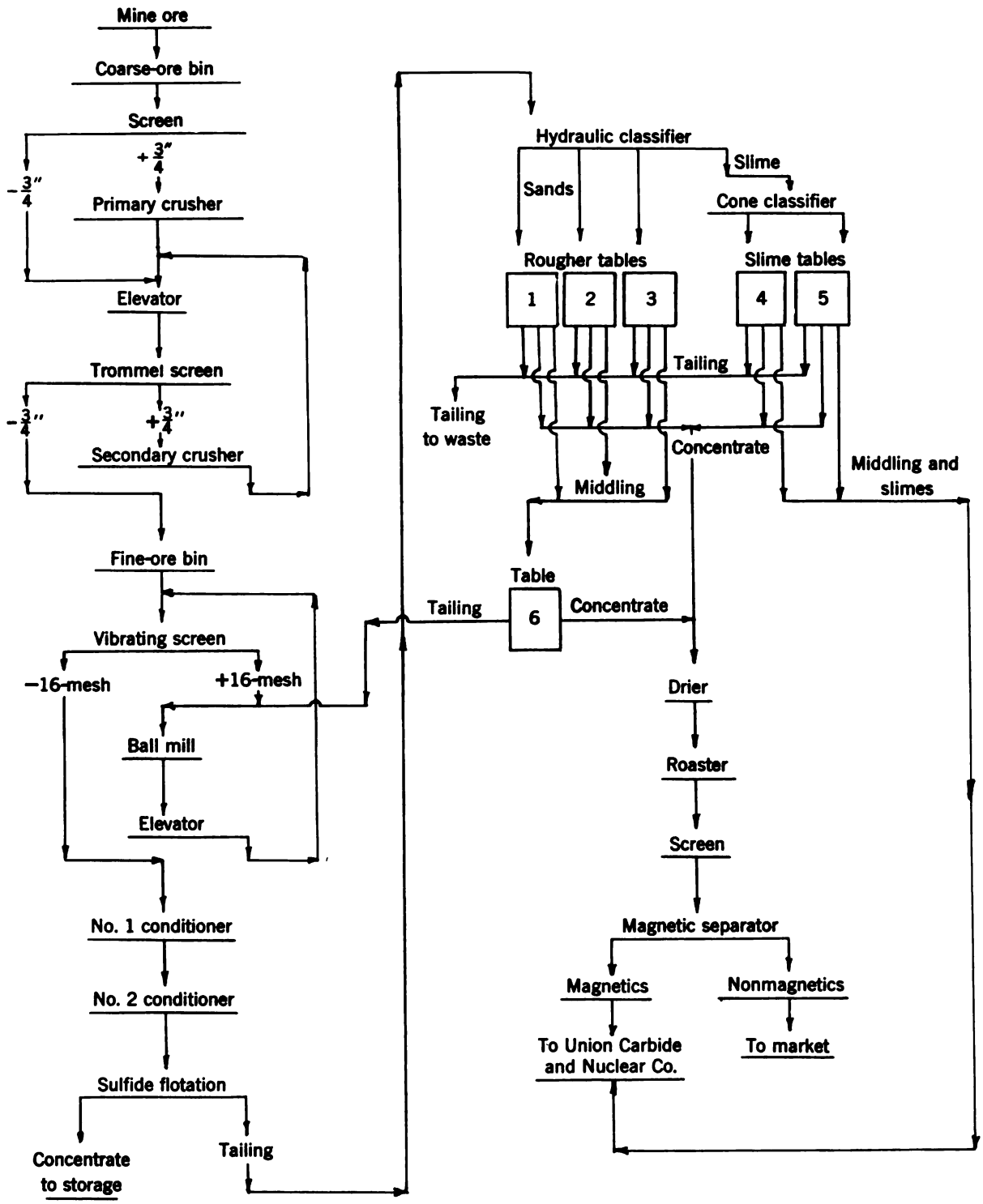

FIGURE 23. - Flowsheet, Round Valley Tungsten Co.

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plus-and-minus-40-mesh products. Each product is treated separately on a three-pole, Stolle-type magnetic separator. The nonmagnetic product is sacked and marketed; and the magnetic product, with the slimes and middlings mentioned above, is shipped to the Union Carbide Nuclear Co. as a byproduct.

New Idria Mining and Chemical Co.

The Strawberry mine and mill, operated by the New Idria Mining and Chemical Co., Fresno, Calif., is approximately 2 miles northwest of Clover Meadow. It is reached by 31 miles of mountain road from Bass Lake, which is 70 miles northeast of Fresno, Calif., on Highway 41. The ore is truck-hauled from three company deposits. The ores in the area are usually tungsten-bearing tactite from contact-metamorphic deposits whose principal minerals are scheelite, garnet, apatite, epidote, molybdenite, powellite, pyrite, pyrrhotite, sericite, quartz, and calcite. The molybdenite content of the ore varies appreciably in the three deposits. The tungsten mineral is concentrated by tabling and flotation, followed by roasting and magnetic separation of the gravity concentrate. The mill treats 125 tons of ore daily, producing a final gravity concentrate containing 68 to 69 percent WO_3 and a flotation concentrate containing 25 to 30 percent WO_3. The average mill feed assays 0.80 percent WO_3, the flotation feed 0.31 percent WO_3, and the combined tailing 0.12 percent WO_3. The flotation feed constitutes 66.5 percent of the total mill feed, and the grind is approximately 3.3 percent plus-100-mesh. Figure 24 is the flowsheet of the milling operations.

Crushing and Grinding

The ores from the three company deposits are truck-hauled to an ore bin near the mill. The ore from the bin is hand-trammed in mine cars to a hopper at the crushing plant; a hand-controlled feeder conveys the ore to a 10- by 20-inch Alloy jaw crusher. The crushed product is fed onto an Overstrom 14- by 36-inch screen. The minus-3/4-inch screen product is conveyed to the fine-ore bin, while the plus-3/4-inch product is fed to a 22-inch, Symons cone crusher and the resulting crushed product conveyed to the fine-ore bin. A vibrating feeder delivers the ore from the fine-ore bins to a bucket elevator, which elevates the ore to a 2- by 6-foot Link Belt vibrating screen, equipped with a 10-mesh screen cloth. The plus-10-mesh product goes to a 5- by 4-foot Eimco ball mill. The ball mill is in closed circuit with the bucket elevator and Link-Belt screen. The minus-10-mesh product is pumped to screen 2, which has an 18-mesh screen cloth. The plus-18-mesh product is returned to the ball mill for regrind, and the minus-18-mesh product goes to a five-spigot Concenco hydraulic classifier. The classifier overflow goes to a 26- by 8-foot thickener.

Gravity Separation

The first three spigot products of the hydraulic classifier are fed separately to the first three rougher tables; the fourth and fifth spigot products are combined and fed to table 4. Table 1 is a Deister diagonal, table 2 a Deister Plato, and tables 3 and 4 Wilfley tables, all 6- by 14-foot. Each table produces a concentrate, a middling, and a tailing. The tailing flows to

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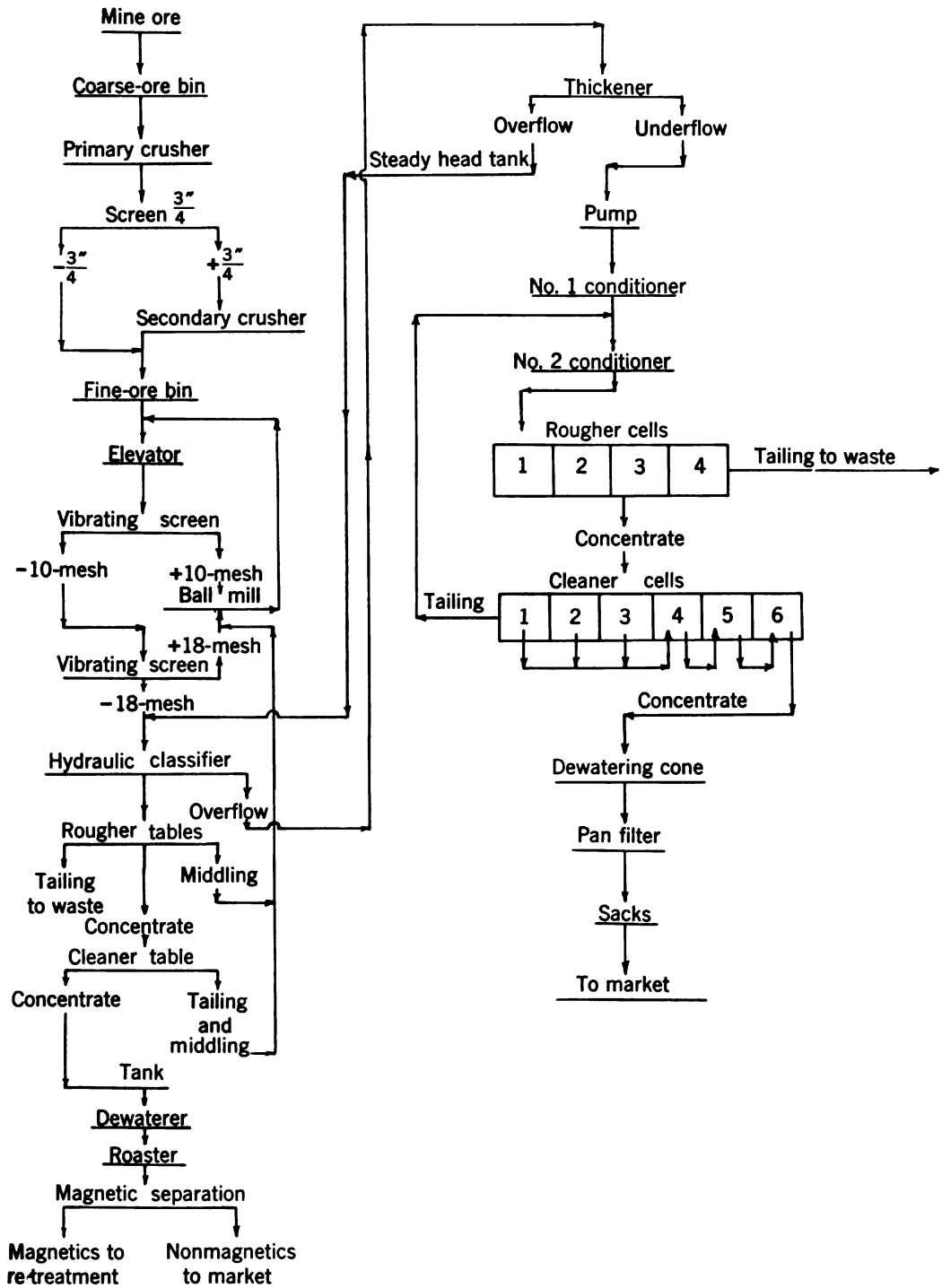

FIGURE 24. - Flowsheet, Strawberry Mill, New Idria Mining & Chemical Co.

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waste, the combined middling to regrind, and the combined concentrate to a Wilfley cleaner table. The cleaner table tailing and middling product are returned to the ball mill for regrind, and the concentrate goes to a tank from which it flows to a dewatering drag in preparation for roasting and magnetic separation.

Flotation

Aeroflox 3000 at the rate of 0.035 pound per ton of solids is added at the overflow discharge of slime and fine sands of the hydraulic spigot classifier. The pulp then flows by gravity to a 26- by 8-foot thickener. The thickener overflow goes to a steadyhead tank and combines with fresh makeup water to supply steady headwater to the hydraulic spigot classifier. The thickener underflow is pumped at 35-percent solids with a 3-inch Denver diaphragm pump to No. 1 conditioner, to which soda ash is added at the rate of 0.96 pound per ton of solids to maintain a pH of 10.1, and 2.32 pounds of sodium silicate per ton of solids. The conditioned pulp then passes to conditioner 2; 0.19 pound of Quebracho, 0.05 pound of Aerosol 18, and 0.63 pound of 1-1 blown tall oil-oleic acid mixture are added per ton of solids. The conditioned pulp then goes to four 36-inch Galigher rougher cells. At the head of the rougher cell is added 0.12 pound of Aerofroth 77; at the third cell is added 0.12 pound of blown tall oil-oleic acid mixture and 0.05 pound of Aerofroth 77 per ton of ore. The rougher tailing goes to waste. The concentrate is fed into the third cell of a bank of six 22-inch Denver cleaner cells; 0.96 pound of sodium silicate per ton of ore is added at the third cell, and 0.04 pound of Quebracho per ton of ore is added at the final cleaner cell. The froth from the first three cleaners is combined and fed to cell 4; the froth from cell 4 is fed to cell 5 and then to cell 6. The cleaner tailing is returned to conditioner 2. The cleaner concentrate goes to a dewatering cone; then to a pan filter. The filtered concentrate containing 8 to 10 percent moisture is sacked in 100-pound bags for shipment. The total reagent cost is approximately 35 cents per ton of ore milled.

Roasting and Magnetic Separation

The roaster is of the rotary-kiln type 30 inches in diameter and 16 feet long, oil-fired. The dewatered table concentrate is dried and roasted while passing through the kiln. A cyclone is provided to collect the flue dust. The fine flue-dust product, which averages 25 to 28 percent WO_3 is shipped "as is." After cooling, the roasted product is fed to a three-pole Dings Wetherill-type magnetic separator. The first pole removes the iron sulfides that are discarded to waste. The second and third poles remove the garnet. The garnet product contains approximately 2 percent WO_3 and is returned to the flotation circuit for further treatment. The scheelite concentrate averages 68 to 69 percent WO_3. This concentrate is sacked in 100-pound bags for market.

Table 31 is a typical analysis of the final table concentrate.

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TABLE 31. - Typical analysis of final table concentrate, Strawberry mine, New Idria Mining and Chemical Co.

Concentrate	Analysis, percent	Concentrate	Analysis, percent
Tungsten trioxide....	66.07	Phosphorus.....	0.03
Tin.....	Trace	Sulfur.....	.14
Copper.....	0.01	Manganese.....	.07
Arsenic.....	None	Lead.....	.01
Antimony.....	None	Zinc.....	.01
Bismuth.....	Trace	Iron.....	.49
Molybdenum.....	1.47		

Tulare County Tungsten Mining Co.

The Tulare mill of the Tulare Tungsten Mining Co. is 18 miles east of Lindsay, Tulare County, Calif. The ore from the mine is truck-hauled 1 mile to the mill. The ore is identified as tactite, containing scheelite, garnet, epidote, calcite, quartz, and minor amounts of sulfides. The scheelite is concentrated by tabling, followed by roasting and magnetic separation of the scheelite concentrate. The mill treats 100 tons of ore per 24 hours. The mill treats 100 tons of ore per 24 hours. The mill feed assays 0.35 to 0.40 percent WO_3. The tailing assays 0.1 percent or less WO_3. Final scheelite concentrate assays 65 to 72 percent WO_3 at 75 to 80 percent recovery. Figure 25 is the flowsheet of the milling operation.

Crushing and Grinding

The ore from the mine is truck-hauled 1 mile to a ramp at the mill and dumped into a 40-ton-capacity coarse-ore bin. A shaking grizzly with 1-inch openings feeds the ore from the bin to a 10- by 20-inch Pacific jaw crusher. The grizzly undersize and the crushed product join at an elevator that discharges the combined products to the fine-ore bin. A belt feeder conveys the ore from the fine-ore bin to a 12- by 36-inch Allis-Chalmers roll. The roll discharges the product to an 8-mesh trommel screen. The screen oversize is fed to a 3- by 4-foot Herman ball mill. The ball-mill discharge and screen undersize flow to a common sump that feeds a 2-inch Wilfley pump, and the combined products are pumped to a four-spigot hydraulic classifier.

Gravity Concentration

The four-spigot products from the hydraulic classifier are fed separately to four Deister Plato tables. The classifier overflow is fed to two slime thickening cones. The underflow from each cone is fed to tables 5 and 6. The overflow from both cones is conserved for mill-wash water.

All the tailing product from table 1 and an 18-inch cut from the tailing of table 2 are returned by an elevator to the ball mill for regrind. Other table tailings are discarded to waste.

An 18-inch cut from the middling product from tables 3, 4, 5, and 6 flows to a 2-inch Wilfley pump and is pumped to the hydraulic classifier for

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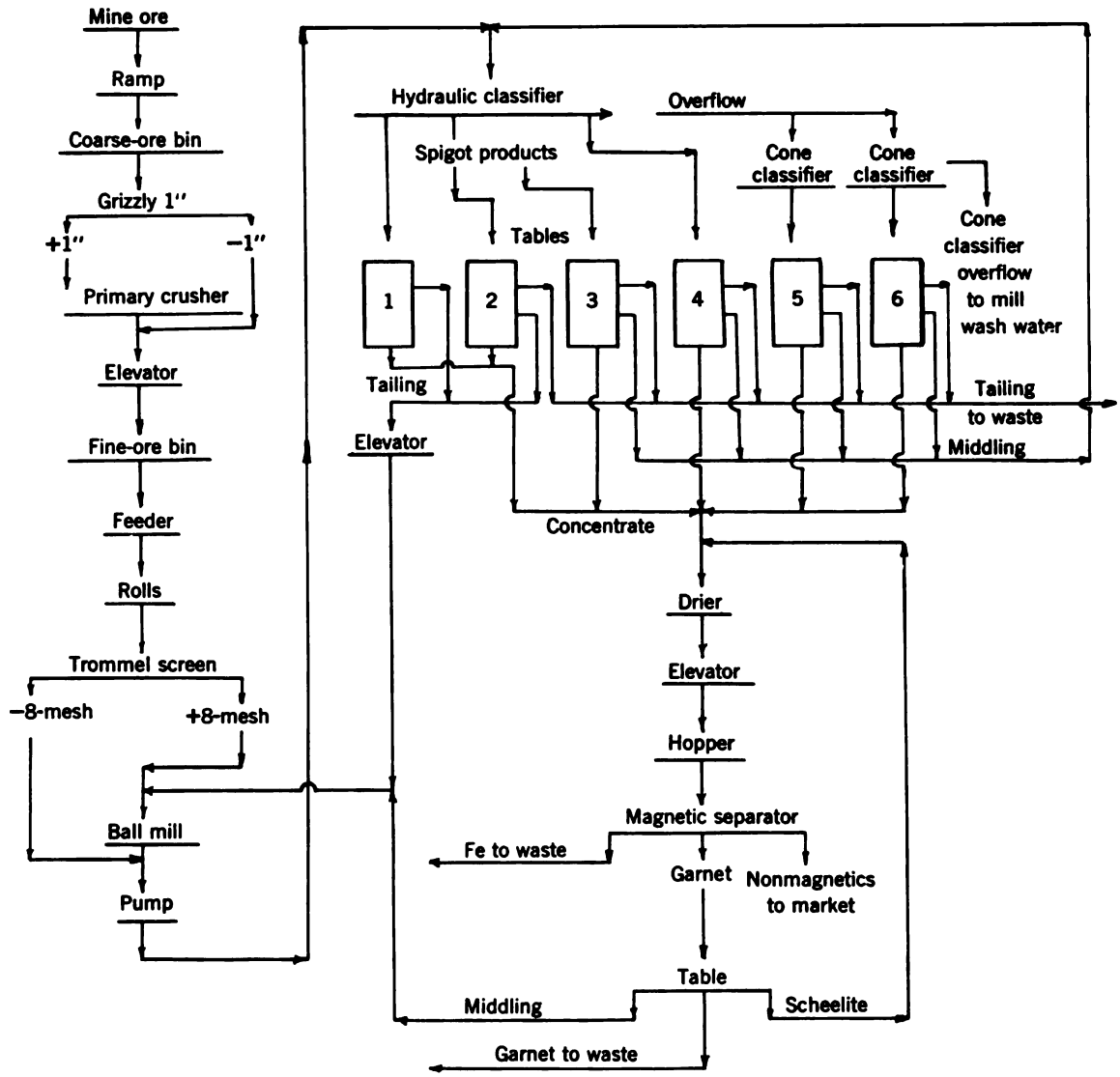

FIGURE 25. - Flowsheet, Tulare Mill, Tulare County Tungsten Mining Co.

recirculation. The concentrate from all six tables is collected in boxes installed at the end of each table, whence the concentrate is shoveled into a wheelbarrow and wheeled to the drying room in preparation for drying, roasting, and magnetic separation.

Magnetic Separation

From the drier the product is fed to an elevator which discharges to a hopper feeding a three-pole Stolle magnetic separator. An iron product is removed at the first pole and garnet at the second and third poles. The scheelite-containing garnet is fed to a small Deister shaking table. The table produces a scheelite concentrate, which is returned to the drier; a middling product, which is returned to the ball mill for regrind; and a garnet concentrate, which is discarded to waste. The nonmagnetic product, assaying 65.0 to 72.0 percent WO_3, is packed in cans and shipped to market.

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Surcease Mining Co.

The Surcease Mining Co. is engaged in mining and two milling operations, plus gravity concentration of hand-picked ore from the washing plant. Its flotation plant (at Red Mountain on Highway 395, San Bernardino County, Calif.) is treating a reserve of 50,000 tons of scheelite-bearing tailings from past operations. Its washing plant, approximately one-half mile east of Atolia in the vicinity of Red Mountain, is treating 300 to 400 cubic yards of open-pit placer and hard rock material daily by jigging the minus-3/4-inch product at the plant site and treating the hand-sorted oversize product at the gravity-concentration plant.

The placer ore and tailings are characterized as quartz-monzonite containing scheelite, magnetite, lime, apatite, and pyrite. The flotation plant treats approximately 140 tons of tailings per 24 hours by gravity and flotation methods, achieving an average recovery of 72 percent. The average tailing mill feed assays 0.32 percent WO_3 and the tailing, 0.09 percent WO_3. More than 70 percent of the values in the millfeed are in the minus-325-mesh product. The magnetite in the flotation and gravity concentrates is recovered by low-intensity magnetic separation. The lime and phosphorus are removed from the concentrates by a two-stage hydrochloric acid leaching process. Some concentrates are shipped to the Molybdenum Corporation of America at Bishop, Calif., for upgrading by high-intensity magnetic separation. Figure 26 is the flowsheet of the milling operations.

Flotation-Plant Tailing Treatment

Grinding and Gravity Concentration

The tailings, loaded into trucks by a skip loader, are hauled 3 miles to the mill hopper. From the hopper the material is fed onto a shaking screen having 3-inch-square openings. The screen oversize is discarded to waste. The undersize is fed to a conveyor belt that delivers the product to a 150-ton-capacity ore bin. A belt feeder delivers the ore from the bin to another conveyor belt, which feeds a 4- by 4-foot Denver ball mill that acts mainly as a repulper. The ball mill discharges to a 42-inch Akins classifier. A 2-inch Wilfley pump delivers the classifier overflow to an 8- by 10-foot No. conditioner. The classifier sands are split into two portions. One-half returns to the ball mill for regrind, and the other half is fed to a Wilfley table that produces a concentrate, a middling, and a tailing. The table middling is returned to the ball mill for regrind; the tailing is fed to a dewatering drag. The dewatering drag overflow is fed to the ball mill, and the sands are discarded to waste. The concentrate, which assays approximately 12 percent WO_3 contains appreciable magnetite and other impurities. The magnetite in the concentrate is recovered by low-intensity magnetic separation and is discarded. The remaining nonmagnetic product is sacked and shipped to the Molybdenum Corporation of America for upgrading to a marketable product by high-intensity magnetic separation.

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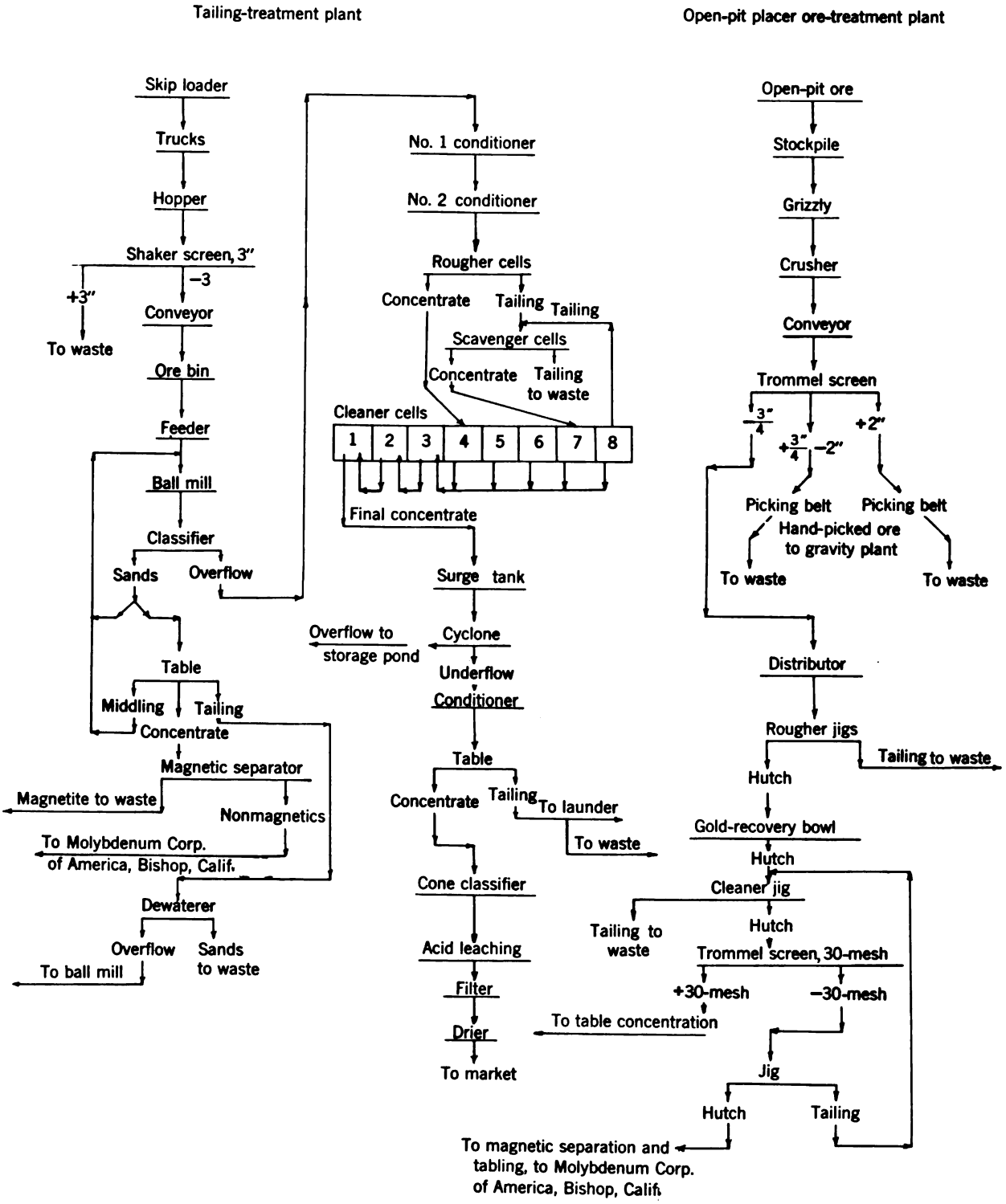

FIGURE 26. - Flowsheet, Tailing and Open-Pit Ore-Treatment Plants, Surcease Mining Co., Red Mountain, Atolia, Calif.

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Flotation

The Akins classifier overflow pulp in conditioner 1 flows by gravity to conditioner 2; the conditioned pulp is fed to two 44-inch Fagergren rougher cells. The rougher tailings are fed to three No. 18 Denver scavenger cells, while the scavenger tails are routed to waste. The concentrates from the rougher and scavenger circuits are fed to eight No. 15 Denver cleaner cells for a three-stage countercurrent cleaning. The rougher concentrate enters the cleaner circuit at cell 4; the scavenger concentrate enters at cell 7. The cleaner tailing is returned to the head of the scavenger circuit. The final concentrate, which assays 3.5 to 4.0 percent WO_3, flows to a 1-inch Wilfley pump that delivers it to a surge tank, from which it flows to another 1-inch Wilfley pump for delivery to a 2-foot porcelain cyclone. The cyclone overflow, used as reclaimed water, flows to a storage pond; the cyclone underflow is fed to a 4- by 6-foot conditioner in which the pulp is conditioned with aluminum sulfate to destroy the effects of flotation reagents before the pulp is tabled. The conditioned pulp is fed to a Dunham Economy shaking table. A 1-inch Wilfley pump delivers the table concentrate to a 2-ton cone in preparation for the acid leach. At this stage the WO_3 content of the concentrate in the cone is between 35.0 and 40.0 percent WO_3, and the tailings between 1.5 and 2.0 percent. Recovery of WO_3 is 50 to 55 percent. The table tailing is routed to waste by way of a specially constructed tailing launder. The tailing launder is 130 feet long, made up of 10-foot sections, 12 inches wide. The bottom of each section is lined with burlap, and the sections are assembled in stepdown fashion. In this manner, approximately 10 percent of the total scheelite in the mill feed is recovered in a product that averages 57.0 percent WO_3. The scheelite product is permitted to accumulate in the bottom of the sectional launders to a depth of one-half inch, which usually requires 5 days. The bottom of the launders is then hand-scraped to recover the scheelite product.

Acid Leaching

The table concentrate flows by gravity from the 2-ton cone classifier to a 6- by 6-foot, rubber-lined, acid-leaching tank for stage leaching in two 3-hour cycles. In the first-stage leaching cycle, hydrochloric acid is added until the pulp reaches a pH of 2.0. This usually requires 1 pound of acid per pound of concentrate. The concentrate is leached for 3 hours to dissolve the calcite. The resulting calcium chloride solution is removed by decantation and the residue given one water wash. In the second-stage leaching cycle, 20-percent hydrochloric acid is added and the concentrate leached 3 hours to dissolve the apatite. The phosphorus-bearing solution is removed by decantation and the residue given five water washes. The washed concentrate product is then transferred to a pan filter and the filtered concentrate product dried in a gas-jet screw drier. After cooling, it is sacked for shipment. The leached product assays 50.0 to 60.0 percent WO_3.

Open-Pit Placer Ore Treatment

The ore from the open-pit deposit is loaded into trucks by power shovels and hauled 1-1/2 miles to the treatment-plant patio stockpile. The stockpiled

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ore is passed over a 16-inch grizzly and the undersize fed to an 18- by 32-inch Telesmith jaw crusher. The oversize product is blasted with dynamite and then passes through the same grizzly. A 24-inch conveyor delivers the crushed product to a 5- by 14-foot trommel screen. The front section of the screen consists of a steel plate with 3/4-inch punched holes. The back section has 2-inch punched holes. The minus-3/4-inch product goes to a 6-inch Wilfley pump and is pumped to a distributing box that feeds four 42- by 42-inch, Pan American, hutch-type, rougher jigs. The plus-3/4-inch, minus-2-inch product goes to a conveyor belt stacker. At the feed end of each conveyor stacker is a dark room equipped with ultraviolet lamps to facilitate the handpicking of scheelite-bearing ore as the material is conveyed to waste at the discharge end of the stacker. The handpicked product is truck-hauled to the gravity concentration plant for further crushing and table concentration.

The minus-3/16-inch hutch product from the rougher jigs is fed to an Ainlay bowl, where small quantities of free gold are recovered. The combined rougher jig tailings go to waste. The gold-free hutch product from the Ainlay bowl flows to a 2-inch Denver vertical pump and is pumped to a 42-inch Pan American hutch-type cleaner jig. The cleaner jig produces a tailing that is discarded to waste and a hutch product that is fed to a 30-mesh trommel screen. The screen oversize product goes to the gravity-concentrating plant for table concentration and the undersize to a pulsating jig. The tailing from the pulsating jig is returned to the head of the cleaner jig, and the hutch product to low-intensity magnetic separation to remove the contained magnetite. Then the product goes to tabling to remove the apatite. The hutch product from the pulsating jig contains an approximate 4 percent WO_3. Low-intensity magnetic separation upgrades the product to 20 percent WO_3; tabling raises it to 40 percent WO_3. The 40-percent concentrate product is then shipped to the Molybdenum Corporation of America plant at Bishop, Calif., for high-intensity magnetic separation treatment to upgrade the product to 60 percent or more WO_3. Table 32 shows the flotation-reagent consumption and place of addition, and table 33 a typical screen analysis of the flotation tailing.

Tungsten Mining Corp.

The Hamme mill of the Tungsten Mining Corp. is 16 miles north of Henderson, Vance County, N. C. The tungsten deposits are hübnerite-quartz veins. They occur as irregular, discontinuous veins or lenticular bodies in the several shear zones. The principal tungsten mineral is hübnerite, which is accompanied by a minor amount of scheelite. Quartz is the dominant gangue mineral, with subordinate amounts of fluorite, rhodochrosite, and sericite. Pyrite is the predominant sulfide mineral found in the ore. It is accompanied by smaller amounts of chalcopyrite, galena, sphalerite, and tetrahedrite. The occurrence of molybdenite is extremely rare. The mill has been expanded to treat 900 tons of ore per 24 hours.

Precautions are taken to minimize production of slimes, because of the friable nature of the tungsten minerals contained in the hard, white quartz matrix. Screening or classification follows each crushing or grinding stage. Briefly, the milling practice is to concentrate the prepared ore by jigging and tabling and to clean the concentrates by sulfide flotation and magnetic

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separation. Accumulated slimes from classifiers and cone classifiers are separately treated on Denver-Buckman tilting-deck tables as described more fully under the heading Slime-Treatment Section. Figure 27 is the flowsheet of the milling operations.

TABLE 32. - Flotation-reagent consumption and place of addition, Surcease Mining Co.

Reagent	Flotation circuit, pounds per ton	Place of addition
Soda ash.....	18.0-20.0	Ball mill
Caustic soda.....	1.0-2.0	Classifier
Sodium silicate.....	3.0	Do.
Sodium silicate.....	2.0	Conditioner 1
Palcotan.....	.3	Do.
Reagent 710.....	.3	Do.
Sodium cyanide.....	.2	Conditioner 2
Oleic acid.....	.7	Do.
Aerosol 18.....	.2	Classifier
Oleic acid.....	.1	No. 1 rougher cell
Reagent 710.....	.2	Do.
Do.	.2	No. 1 scavenger cell
Reagent	**Table circuit, pounds per ton of concentrate**	**Place of addition**
Aluminum sulfate.....	30.0	Table conditioner
Palcotan.....	.5	Do.
Sodium silicate.....	.25	Do.

TABLE 33. - Typical screen analysis of flotation tailing, Surcease Mining Co.

Mesh	Weight-percent	Analysis, percent WO_3	Distribution of WO_3, percent
65.....	3.7	0.071	3.6
100.....	9.1	.050	6.2
150.....	9.2	.050	6.3
200.....	8.6	.028	3.3
+325.....	15.9	.014	3.1
-325.....	53.5	.106	77.5

Crushing Section

Ten-ton-capacity Ford trucks deliver the ore from the mine to a 150-ton steel bin, equipped with a 12-inch grizzly. The bin discharges through a single opening controlled by a No. 5 Ross chain feeder onto a 36-inch conveyor belt equipped with a Dings electromagnetic head pulley to remove tramp iron. The conveyor belt discharges onto a short-sloping, 1-3/4-inch grizzly; the fines drop to a 30-inch conveyor belt. The oversize product falls into an 18- by 36-inch Birdsboro-Buchanan jaw crusher set at 1-3/4 inches. The crushed product joins the grizzly fines on the 30-inch conveyor. The 30-inch conveyor

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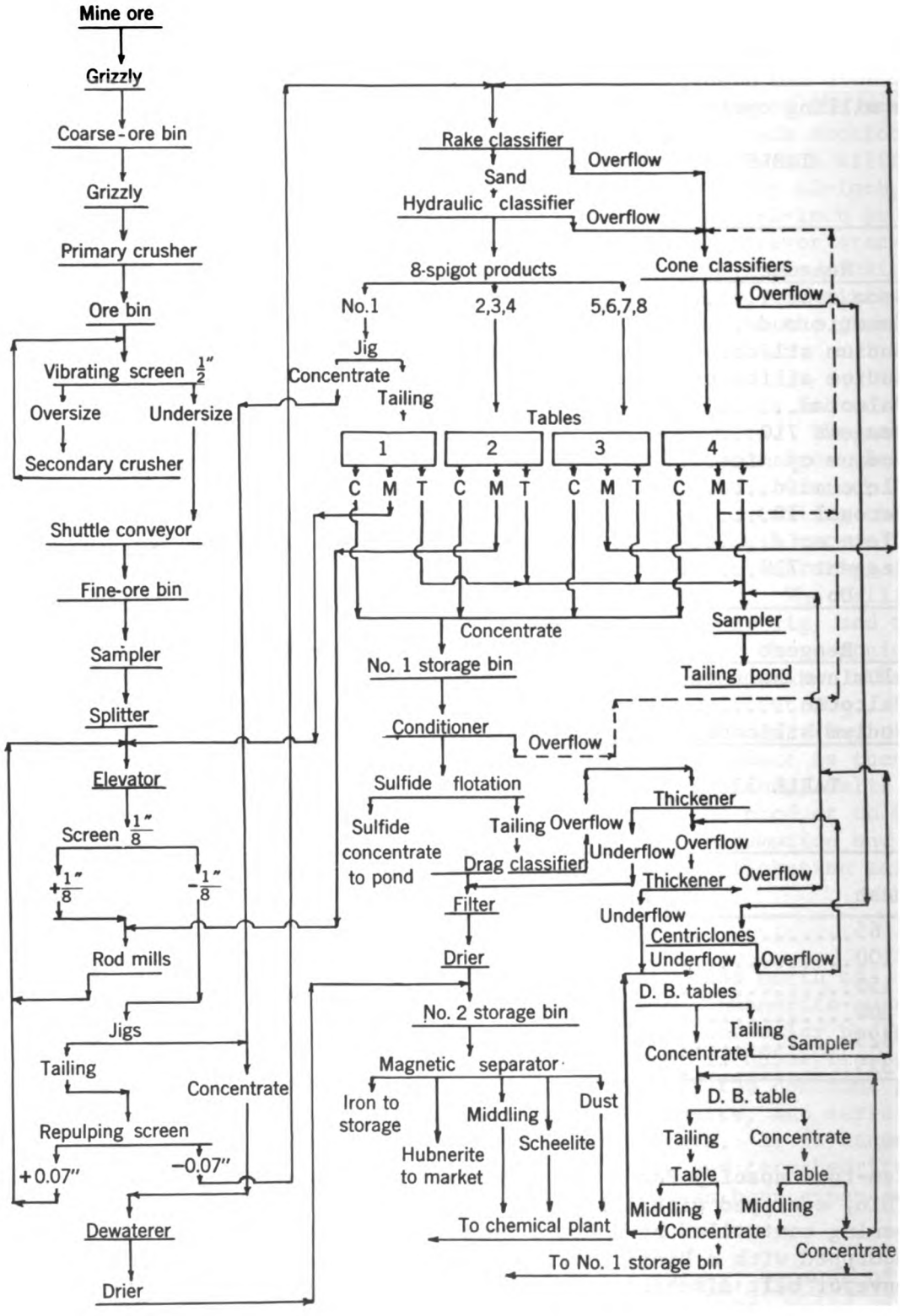

FIGURE 27. - Flowsheet, Hamme Mill, Tungsten Mining Corp.

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discharges the combined products to a 300-ton, circular, steel storage bin. The 300-ton storage bin is discharged through four 16-inch square openings by two double-acting reciprocating pan feeders. Only one feeder is normally in operation at a time. Ore from these feeders is discharged onto a 24-inch horizontal conveyor that in turn discharges onto a 20-inch incline conveyor. This conveyor transfers the ore to two additional, 22-inch, inclined conveyors that deliver the ore to a 26- by 60-inch Robins-Eliptex vibrating screen having 1/2- by 1/2-inch screen openings. The plus-1/2-inch screen product is fed, through a chute, to a 3-foot, short-head, coarse-bowl, Symons cone crusher, set at 3/8-inch. The crusher discharge is returned to the Robins-Eliptex vibrating screen by the same two 22-inch conveyors that feed the ore from the 300-ton ore bin to the Robins-Eliptex screen. The minus-1/2-inch screen product goes to a shuttle conveyor operating over a 600-ton-capacity, suspension bunker fine-ore bin by two 18-inch inclined conveyors operating in series. The shuttle conveyor travels on a track with automatic reversing equipment at each end of the 46-foot-long bin. Automatic reversal of the conveyor belt also takes place concurrently with reversal of the carriage on the track. In this manner, uniform distribution of the ore is attained over the entire length of the bin.

Gravity-Concentrating Section

The ore in the 600-ton fine-ore bin is fed uniformly to two parallel grinding and sand-concentrating circuits by 11 Stephens-Adamson 12- by 16-inch, open-type, rotary pan feeders spaced equally along the 46-foot length of the bin. Two conveyors carry the ore to a common chute under the center of the fine-ore bin, where it is transferred to a 20-inch inclined conveyor belt. The inclined conveyor belt is equipped with a Merrick-type S weightometer.

A Geary-Jennings automatic sampler, at the head pulley of the weightometer belt, cuts a sample of the mill feed every 2 minutes. The sample is prepared in a small sample mill consisting of a pan feeder, three small conveyors, a 4- by 6-inch laboratory jaw crusher, a set of 10- by 6-inch laboratory rolls, a 5-inch-belt bucket elevator, a 14- by 36-inch Stephens-Adamson vibrating screen with an 0.11-inch screen opening, and another Geary-Jennings sampler. The final sample weighs about 25 pounds.

Following the first automatic sampler, a splitter cuts the mill feed into two equal parts; one part for each parallel circuit. Each part is fed to an 18-inch-belt bucket elevator by a 20-inch belt conveyor. Each elevator has a 40-foot lift and discharges onto two 4- by 4-foot Hum-mer vibrating screens with 1/8-inch-opening screen sections.

The plus-1/8-inch product flows by gravity to two 4- by 6-foot Marcy and peripheral discharge rod mills that are in closed circuit with the two Hum-mer screens mentioned above by way of the two 18-inch-belt bucket elevators. The minus-1/8-inch product flows by gravity to four 36- by 36-inch Bendelari jigs operated in parallel. Approximately 40 to 45 percent of the total tungsten-mineral recovery in the mill is made by these jigs. The dewatered jig hutch and bed products are fed into a screw-conveyor heat-lamp drier. The drier is equipped with eighty 375-watt lamps and can dry 0.3 ton of concentrate per

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hour. The dried product is discharged into a bucket elevator, which raises it to a steel storage bin for magnetic separation.

The jig tailing flows by gravity to two 5-inch Wilfley sand pumps, which discharge onto two-pan repulping 4- by 8-foot Hum-mer vibrating screens that have 0.07-inch-opening screen sections. The screen oversize flows to the rod mills. The ground pulp from the rod mills flows by gravity to the 18-inch-belt bucket elevators that return it to the 1/8-inch screen.

The minus-0.07-inch product flows by gravity to two 4-inch Wilfley sand pumps, discharge into two 4- by 25-foot Denver rake classifiers. The rake-classifier sands flow by gravity to two eight-spigot Concenco hydraulic classifiers. The first coarse-sand spigot products flow to two 12- by 12-inch Crangle pulsating jigs. The jig tailing flows by gravity to the first sand tables of the two sand sections. The Crangle-jig concentrate is combined with the Bendelari-jig concentrate for dewatering, drying, storage, and magnetic separation. The remaining 14-spigot products of the two hydraulic classifiers flow by gravity to 14 sand concentrating tables that comprise 11 Deister No. 6, 2 Wilfley No. 6, and 1 Wemco table.

The first sand table of each section produces a concentrate that is pumped to an 8÷ by 8-foot steel storage bin in preparation for the sulfide flotation, a sulfide middling that is returned by gravity to the 18-inch belt bucket elevator feeding the screen and the Bendelari jig, a sand middling that is pumped to the rod mill for regrind, and a tailing that is sampled and discarded to waste.

The remaining four sand tables of each section produce a concentrate (which is pumped to the sulfide-flotation storage bin), a middling (which is pumped to the rake classifier), and a tailing (which will join the other table tailings now discarded to waste).

The combined overflow of the rake and hydraulic classifiers is fed to four 8-foot cone classifiers operated in parallel. Each cone underflow is fed to a Deister No. 6 diagonal deck concentrating table. The table concentrate is pumped to the sulfide-flotation storage bin. The middling is returned to the rake classifier in both sections, and the tailing joins other tailings for disposal to waste.

Slime-Treatment Section

The overflow from the four 8-foot cone classifiers of both sand sections unites and goes to three 20-inch centriclones. The centriclones underflow contains approximately 90 percent of the plus-15-micron size particles. They are fed to eight Denver-Buckman tilting-deck tables. Each table has five 6- by 6-foot decks covered with flat, dimpled-rubber covers. The tables are fed with the decks at a low angle of slope for about 6 minutes; the decks then tilt to a steep slope in the opposite direction. At that time the decks are automatically washed clean with water sprays. The washing time is approximately 45 seconds. The decks are then returned to their normal position, and the cycle is repeated.

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The tailings from the eight Denver-Buckman tables flow by gravity to a Geary-Jennings automatic sampler, thence to the main tailing launder through the main sampler and to waste. The combined concentrate from these tables is pumped to a 10-foot cone. The cone overflow is returned to the centriclones and the underflow pumped to two Denver-Buckman cleaner tables, which produce a concentrate to be upgraded on a Deister No. 6 slime table. The concentrate thus produced goes to the sulfide-flotation storage bin. The tailing from the Deister table is returned to the cleaner tables, and the tailing from the cleaner tables is recleaned on two Deister No. 6 slime tables that produce a concentrate for sulfide flotation, a middling for recleaning, and a tailing which is returned to the eight rougher tables. The centriclone overflow, which consists largely of minus-15-micron particles, flows by gravity to a 30-foot thickener and thence to a 24-foot thickener of equal volume, for further removal of solids that are pumped to the eight Denver-Buckman tilting-deck tables to join the centriclones' underflow.

Sulfide-Flotation Section

The sulfide-flotation circuit includes a 24-hour capacity steel storage bin, a 3- by 3-foot conditioner, and four Denver No. 18 Sub-A flotation cells. The combined gravity concentrate stored in the bin normally assays 25 to 30 percent sulfur and 20 to 25 percent WO_3. The sulfide concentrate product contains about 0.05 percent WO_3, and the sulfide flotation tailing (tungsten concentrate) 45 to 50 percent WO_3 and about 3 percent sulfur. Following the flotation of sulfide minerals, the tungsten-concentrate product is dewatered in an Esperanza-type classifier, the overflow of which is pumped to a 9- by 6-foot thickener. The thickener underflow is fed to a 3- by 6-foot Dorrco filter, along with the dewatered concentrate from the Esperanza classifier. The thickener overflow goes to another 24-foot thickener.

The filter cake discharges into a duplex-screw conveyor drier equipped with eighty 375-watt-each heat lamps suspended directly above the screws. The dried concentrate is discharged into the same elevator used for the jig concentrate and stored in a steel bin for magnetic separation.

Magnetic Separation Section

The combined concentrates in the steel storage bin are fed to a 2- by 2-foot, double-deck, Hum-mer vibrating screen equipped with a 0.09-square-inch opening screen section and a 0.02-square-inch opening screen section. The two undersize products go to separate storage bins that feed the magnetic separator; the oversize product is fed to an 8- by 5-inch laboratory roll in closed circuit with the Hum-mer screen.

The magnetic separator is a Dings six-pole, high-intensity, crossbelt-type machine. The first crossbelt removes an iron product, largely metallic iron, that assays about 5.0 percent WO_3. This product is now stored pending development of a suitable process for upgrading it.

The second crossbelt removes a middling product assaying 15 to 20 percent WO_3 and containing some metallic iron, sulfides, rhodochrosite, and hübnerite.

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The third, fourth, fifth, and sixth crossbelts remove a high-grade hubnerite concentrate assaying 70 to 73 percent WO_3. The end product, which is nonmagnetic, contains scheelite, fluorite, and sulfides that assay 15 to 20 percent WO_3.

A Pangborn bag-type dust collector is used in conjunction with the magnetic separator and the drying circuit. The dust collected from these circuits assays approximately 45.0 percent WO_3. The scheelite, middling, and dust products from the magnetic separator are processed in the chemical plant producing synthetic scheelite. The chemical plant began operating on March 15, 1955. It produces a high-grade sintered scheelite concentrate. Details of the chemical plant are not available for publication.

Table 34 shows the concentrate distribution and table 35 the heat-lamp drier costs.

TABLE 34. - Concentrate distribution, Tungsten Mining Corp.

Product	Weight-percent	WO_3, percent	Distribution WO_3, percent
All-jig concentrate..	37.0	63.0	44.0
Table concentration and flotation.....	63.0	46.7	56.0
Total.....	100.0	109.7	100.0

TABLE 35. - Heat-lamp drier operating cost, Tungsten Mining Corp.

Power (lamps)...	$1.33	Labor (operation and maintenance)[1]...	$0.15
Power (drive)...	.03		
Supplies.....	.14	Total cost per ton of concentrate dried....	[1] 1.65

[1] Estimated cost.

Minerals Engineering Co.; Montana Tungsten Division

The Glen mill of the Montana Tungsten Division of the Minerals Engineering Co. is 23 miles north of Dillon, Beaverhead County, Mont. The mine is 9-1/2 miles northwest of the mill. The ore is truck-hauled from the mine to the mill by ten 18-ton trucks. The ore occurs along a contact of limestone and hornfels with a quartz-monazite known as the Pioneer batholith. The mineralized zone referred to as tactite results from the metasomatic effects of the intrusive and is composed chiefly of garnet. The principal tungsten mineral is an impure, fine-grained scheelite disseminated throughout the tactite. Other minerals present are minor amounts of powellite, molybdenite, chalcopyrite, azurite, malachite, and garnet. Other gangue minerals are calcite, orthoclase, garnet, chalcedony, and quartz.

The mill treats an average of 900 tons of ore per 24 hours, using flotation and tabling methods. The mill water is pumped 12,000 feet from the

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Big Hole River by two-stage pumping; there are two pumps at the river and two at the booster station. The total lift is approximately 500 feet. During the cold winter weather the mill-circuit water is heated to 65° to 70° F. and fed to the ball mills.

The average mill feed assays 0.35 to 0.40 percent WO_3, the tailing 0.035 percent WO_3. The average recovery is 92.0 percent. The mill produces high- and low-grade products. The high-grade products, which assay 55 to 60 percent WO_3, are shipped for upgrading to the Salt Lake Tungsten Co. at Salt Lake City, Utah, owned by Mineral Engineering Co. An acid leach removes the copper and sulfur. The low-grade product, assaying approximately 10 percent WO_3, is also shipped to the Salt Lake Tungsten Co. chemical plant for pressure digestion and conversion to synthetic scheelite. The chemical plant section of the Salt Lake Tungsten Co. is owned jointly by the Minerals Engineering Co. and the Sylvania Electric Corp. Figure 28 is the flowsheet of the milling operations.

Crushing and Grinding Section

Ore from the open-pit mine is truck-hauled to the crushing plant in 18-ton-capacity trucks and dumped on a Traylor apron feeder or stockpiled near the crushing plant. The apron feeder delivers the ore to a 24- by 36-inch Rogers jaw crusher for reduction to 3 inches. The crushed product is conveyed to a surge bin from which it is conveyed to a 4- by 10-foot Allis-Chalmers vibrating screen with 3/4-inch openings. A stacker conveyor feeds the screen oversize to a 4-foot Symons short-head cone crusher set at 5/8 inch. The crushed product and the screen undersize are combined and conveyed to an open-air stockpile.

Under the stockpile are two concrete tunnels housing a conveyor system that feeds the ball mills of two parallel circuits. Circuit 1 consists of a 7- by 8-foot Traylor ball mill in closed circuit with a 60-inch Akins classifier. Circuit 2 consists of a Marcy 75 ball mill in closed circuit with a 48-inch Akins classifier. The overflow from each classifier is maintained at 30 to 35 percent solids. The grind is between 47 and 50 percent minus-200-mesh, with 3 percent on 65-mesh.

Concentration

The classifier overflow from circuit 1 flows by gravity to a No. 24 Denver unit cell for the primary roughing stage. The unit cell produces a concentrate and tailing. The concentrate is fed to No. 1 Deister table for the final cleaning. The tailing is fed to a 10-cell bank of No. 18 MS roughers, with the feed entering the bank at cell 2. The concentrate from the first three rougher cells is combined and routed to No. 1 Deister table. The concentrate from the remaining seven cells is combined and recirculated to cell 1. The rougher tailing flows to an eight-cell bank of No. 24 Denver scavenger cells, the feed entering the bank of cells at cell 1. The concentrate from the last seven scavenger cells is combined and recirculated to scavenger cell 1, whereas the concentrate from cell 1 is routed to No. 1 Deister table for cleaning. The scavenger-circuit tailing flows to waste

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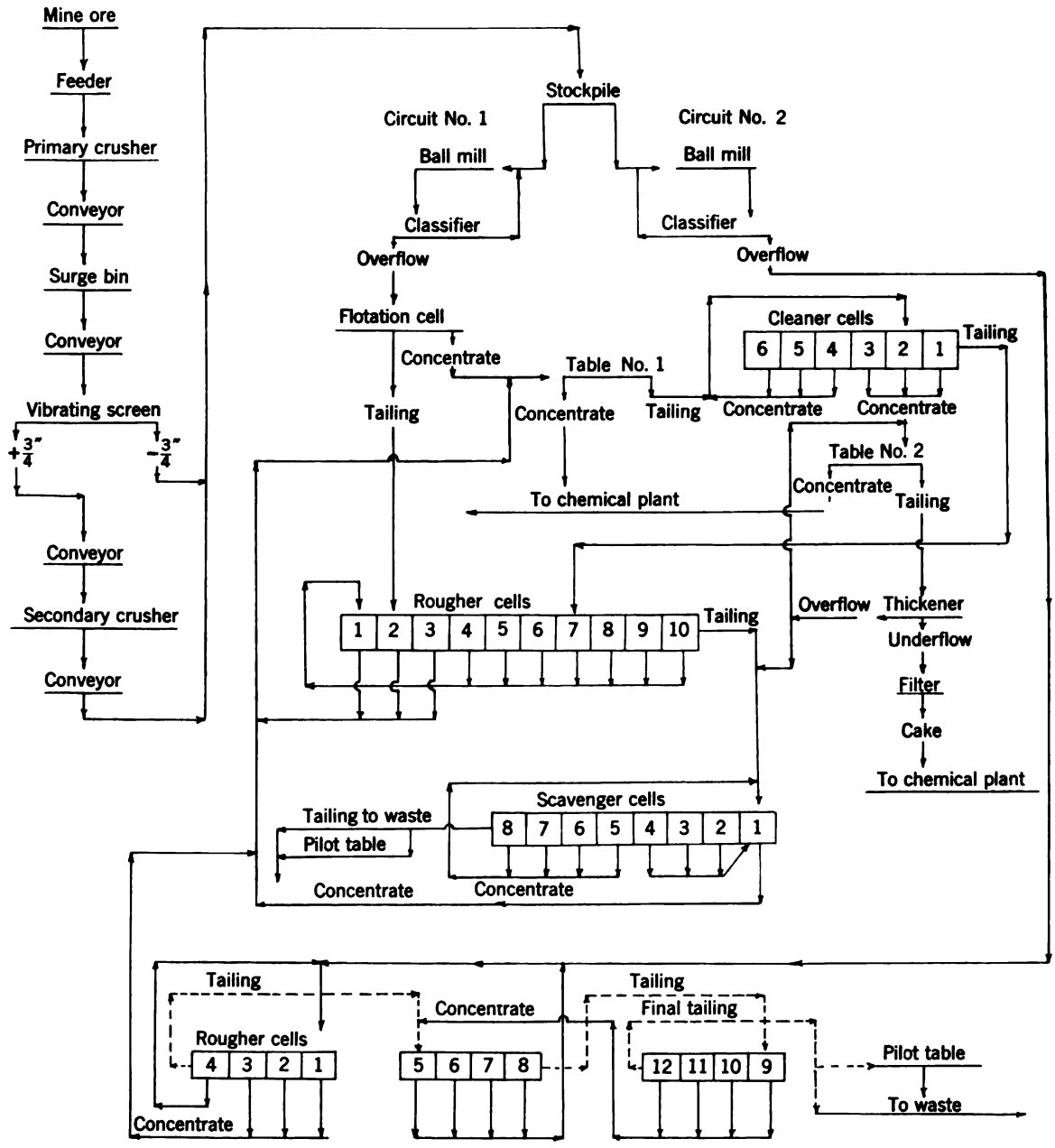

FIGURE 28. - Flowsheet, Glen Mill, Montana Tungsten Division, Minerals Engineering Co.

after a representative portion of the tailing is passed over a pilot table for visual observation and plant control. Deister table 1 produces a final concentrate and a tailing. The tailing flows to a 2-inch Wilfley pump and is pumped to a six-cell bank of No. 18 MS cleaners; the feed enters at cell 2. The concentrate from the first three cleaner cells flows to Deister slime table 2. The concentrate from the remaining three cleaner cells is combined and recirculated to cleaner cell 2, while the cleaner tailing is routed to cell 7 of the MS rougher circuit.

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The Deister slime table produces a concentrate and tailing. The concentrate, which assays 55 to 60 percent WO_3, is placed in 20-gallon cans and shipped to the company upgrading plant at Salt Lake City for acid-leaching treatment to eliminate copper and sulfur from the concentrate.

The slime-table tailing flows to a 12- by 20-foot thickener. The thickener underflow is pumped to a 3-foot Eimco disk filter. The filter cake, which assays about 10.0 percent WO_3 and contains about 18 percent moisture, is bulk-loaded into railroad-automobile freight cars and shipped to the Salt Lake Tungsten Co. for the pressure-digestion treatment and conversion to synthetic scheelite.

Part of the thickener overflow is utilized as makeup water in the scavenger circuit; the remainder is used as wash water for the Deister slime table.

The classifier overflow from circuit 2 flows to three sections of rougher cells; each section is made up of four No. 24 Denver-Buckman Sub-A cells, the feed entering cell 1 of the first section. The concentrate from the first three cells of section 1 is combined and routed to Deister table 1 of circuit 1. The concentrate from cell 4 of the first section is recirculated to cell 1. Tailing from this section goes to cell 5 of the second section. The concentrates from all four cells of the second section are combined and pumped to cell 1 of the first section, and the tailing goes to cell 9 of the third section. Concentrates from all four cells of the third section are combined and pumped to cell 5 of section 2. The final tailing emerging from section 3 is discarded to waste after a representative portion is passed over a pilot table for visual observation and circuit control. Table 36 shows the consumption of reagent and place of addition.

TABLE 36. - Flotation-reagent consumption and place of addition, Minerals Engineering Co.

Reagent	Pounds per ton	Place of addition
Soda ash.....	3.0-5.0	Ball mill
Aliphat 44-E.....	1.50	Staged to rougher circuit
Sodium silicate.....	3.00	Staged to rougher circuit and cleaners
Pine oil.....	.15-0.18	Rougher circuit

Salt Lake Tungsten Co. Synthetic Scheelite Chemical Plant

The Salt Lake Tungsten Co. custom chemical plant treats low-grade tungsten concentrates to produce synthetic scheelite. The plant, at 2160 Indiana Ave., Salt Lake City, Utah, is owned jointly by the Sylvania Electric Products, Inc., and the Minerals Engineering Co.

The plant treats low-grade concentrate from the Mineral Engineering Co., Montana Tungsten Division, at Glen, Mont., and from mines in Utah, Nevada, Colorado, and Arizona and other sources. Approximately 50 tons of low-grade concentrates is treated daily.

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The process used is based on decomposition of the finely ground scheelite concentrate with sodium carbonate at an elevated temperature and pressure in reactors to produce a sodium tungstate and final precipitation of the sodium tungstate from the solution with calcium chloride to produce a synthetic scheelite - $CaWO_4$. Figure 29 is the flowsheet of the Salt Lake Tungsten Co. synthetic scheelite chemical plant.

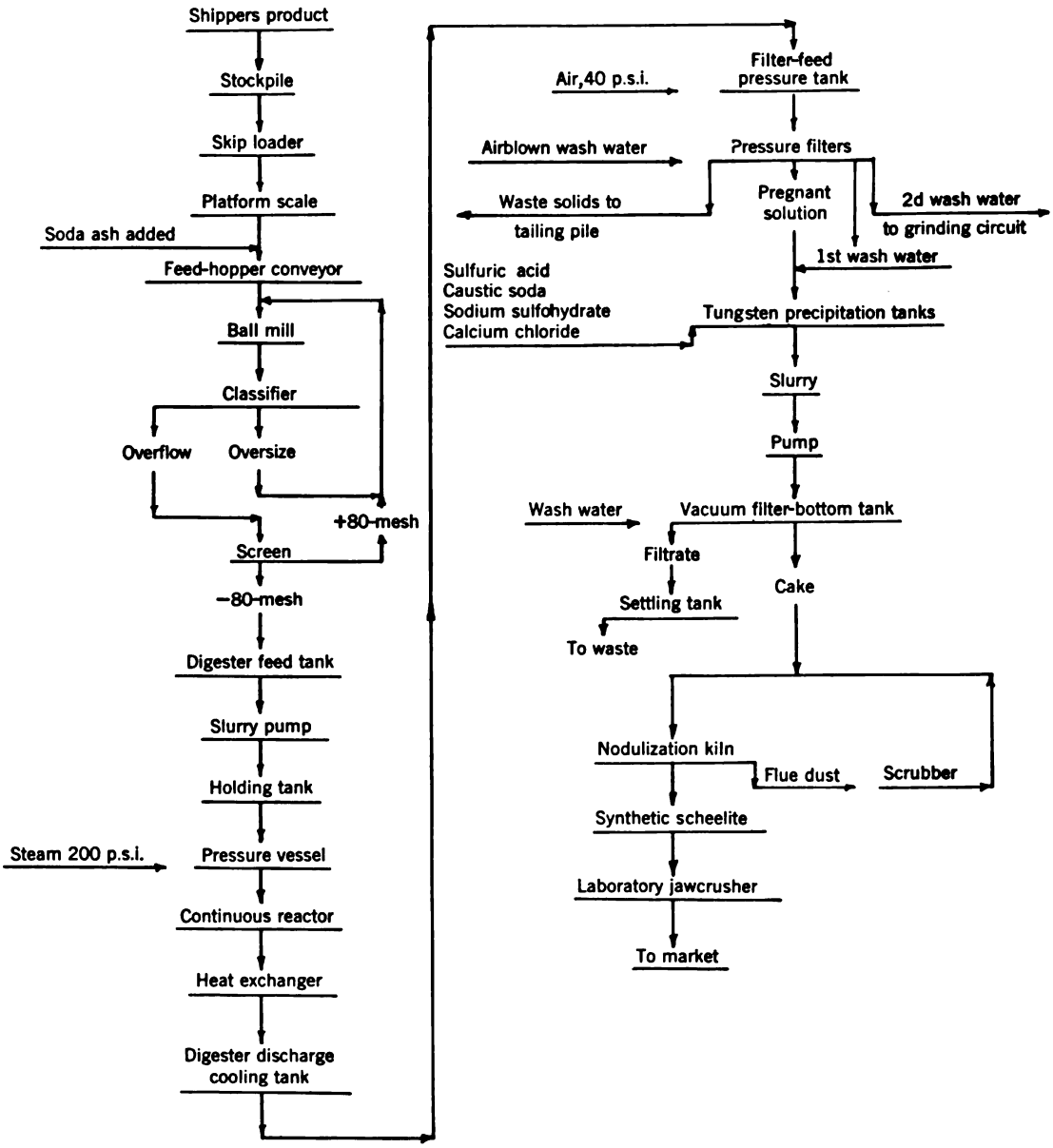

FIGURE 29. - Flowsheet, Salt Lake Tungsten Co. Synthetic Scheelite Chemical Plant.

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Plant Operation

Preparation of Tungsten Concentrate for Treatment

Tungsten concentrates from various shippers are received by rail or truck, weighed, sampled, and stored individually on a concrete patio stockpile. These individual lot shipments of concentrates assay 5 to 15 percent WO_3. A skip loader blends the various lots of concentrates to give a uniform feed, usually averaging 7 percent WO_3. The skip loader then loads the blended concentrate into a feed hopper. Simultaneously, a predetermined amount of sodium carbonate is added into the feed hopper, and the mixture is conveyed to a Hardy ball mill for repulping and grinding. Most of the repulped material will pass a 100-mesh screen. Any coarse scheelite present is ground to minus-80-mesh. The Hardy ball mill is in closed circuit with a Hardinge countercurrent classifier and a Wemco screen having 80-mesh openings. The classifier overflow is fed to the Wemco screen. The classifier and screen oversize are returned to the ball mill for regrind, and the screen undersize flows to the digester feed tank. Wash water resulting from the plant's filtering operations is added to the ball mill to produce a slurry containing about 50 percent solids and 7 percent WO_3. The prepared slurry is then pumped to a holding tank, from which it flows by gravity to a pressure vessel. High-pressure steam is then introduced into the pressure vessel to force the charge into the reactors.

Digestion in Continuous Pressure Reactors

The continuous pressure reactors used at the Salt Lake plant are made up of a group of six stationary vertical pressure vessels arranged in a circle and connected in series, through which the slurry passes in a continuous flow. The first pressure vessel in which the feed enters is 24 inches in diameter and 24 feet in height. The remaining five pressure vessels are 18 inches in diameter and 24 feet in height. All six pressure vessels are surrounded by a single housing to facilitate external auxiliary gas heating.

Steam is introduced, and the temperature is increased until a reacting temperature of 370° F. and a pressure of 200 p.s.i. are reached. According to Blair T. Burwell (6) the reaction, as carried out in the pressure vessel, usually is given as:

$$CaWO_4 + Na_2CO_3 \xrightarrow{\text{Heat}} Na_2WO_4 + CaCO_3.$$

However, the mechanism of the reaction is probably first

$$2Na_2CO_3 + H_2O \xrightarrow{\text{Heat}} 2NaOH + Na_2CO_3 + CO_2.$$

In the second equation the ratio of NaOH to Na_2CO_3 is a function of concentration, heat, and pressure. In a closed vessel, the digestion is probably effected by the concentration of the caustic soda; the CO_2 recombines with the caustic soda during cooling to form carbonate and bicarbonate. The process is best adapted to scheelite ores but can be used for other tungsten minerals by employing a caustic soda solution.

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Application of heat and pressure to the abrasive slurry creates mechanical problems. The temperature and concentration of sodium hydroxide and sodium tungstate are also critical. At the Salt Lake City plant automatic temperature controls are used on the reacting vessels to maintain high reacting efficiency. The rate of conversion of $CaWO_4$ to Na_2WO_4 decreases with increasing concentration of sodium tungstate which limits the process to low-grade material, under 15 percent WO_3 content. Higher grade concentrates require multiple treatment.

When the reaction proceeds at a temperature below the critical levels required to convert fluorine, tine, and other impurities to soluble sodium salts, the chemical attack is selective on tungsten and molybdenum minerals. If proper control of the pressure reaction is maintained, arsenic, antimony, and some phosphorus minerals are partly attacked; lead, copper, and other minerals are not affected however.

Pressure Filtration

The digested material or slurry from the reactors flows through a series of cooling coils or heat exchangers and into a surge tank of approximately 28-ton capacity. From the surge tank the pulp flows to the filter-feed pressure tank, where it is subjected to about 40 p.s.i. air pressure to force the pulp to either one of the two Eimco-Burwell pressure filters. Each filter unit consists of 30 circular frames, each about 24 inches in diameter and 2 inches in width, mounted vertically on a horizontal revolving shaft. The shaft rotates the frames between 31 expandable plates at the start and end of each cycle. The filter cloths are permanently mounted on the plates with one strand of medium-heavy wire. When the frames are in position, the plates are closed on the frames, with air at 65 p.s.i. pressure. The material to be filtered is introduced into the bottom of the sealed frames by an automatically controlled valve. Loading of frames is metered to produce a separate cake on each filter surface with a division or "split" between the cakes.

Following the feed cycle, high-pressure air is introduced through the same feed channel into the frames to blow out the pregnant tungsten-bearing solution from the cake, leaving a fairly dry cake. Two water washes are then introduced in the same manner, followed by a blow cycle to produce a final cake containing very low soluble values. The residual cake usually contains less than 15 percent moisture, which is about 30 percent less than can be obtained on a vacuum filter. Recovery of soluble tungstate exceeds 99.8 percent when 1 ton of water is used per ton of feed. The filter cycle is usually of 23- to 30-minute duration, with the filtering capacity of each unit ranging between 30 and 35 tons per day.

In the filtering process the first wash water is combined with the pregnant solution and flows to the tungsten-precipitating tanks. The second wash is utilized in the grinding or repulping circuit. The residual cakes in the filter frames are dumped on a concrete floor immediately below the filter assembly, where they are picked up with a front-end loader and carted to waste.

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Tungsten Precipitation From Pregnant Solution

Sulfuric acid to eliminate CO_2 is added to the pregnant solution in the tungsten-precipitation tanks. The pH is raised to 8.5 with caustic soda. Sodium sulfohydrate and calcium chloride are then added to precipitate the tungsten as $CaWO_4$. The resulting slurry is then pumped to a vacuum filter-bottom tank. The filtrate from this tank flows to a settling tank and thence to waste. The remaining cake or precipitate in the filter bottom tank is washed with water until the soluble sulfate present is washed out of the precipitate. The washed cake is removed from the filter tank by hand shoveling and routed to the nodulizing kiln.

Nodulization

Nodulization of the filter cake is conducted in a 24-foot by 32-inch gas-fired rotary kiln heated to 1,000° C. Flue dust is recovered in a scrubber and returned to the kiln. The nodulized synthetic scheelite, which assays 67 to 68 percent WO_3 is passed through a laboratory crusher and reduced to a 1/4- or 3/8-inch-size product. The crushed product is then sampled and packed in 350-pound net weight drums for shipment to market.

PROCESSING TUNGSTEN

The high melting point of tungsten (3,419° C.), its reactivity, and the nonavailability of refractories that can be used to contain it in the molten state preclude the use of conventional reduction and melting techniques. The production of tungsten metal can be conveniently considered under four headings: (1) Decomposition of the ore or concentrate, (2) preparation of the pure oxide, (3) reduction to metal powder, and (4) conversion of the metal powder to massive metal.

Production of Metallic Tungsten

Decomposition of Ore and Concentrate

The ore or concentrate used for preparing pure tungsten must be comparatively free of impurities, although most of the pure metal is made from wolframite (2, pp. 1002-1007). Some plants, of somewhat different design, use scheelite as a starting material.

Various methods are used to decompose the tungsten ores or concentrate. In one method the ore or concentrate is fused with sodium carbonate to form a water-soluble tungstate. About 2 hours at 800° C. is required for a charge of about 200 pounds of 70 percent WO_3 concentrate and 100 pounds of sodium carbonate. The cooled mass is crushed to pea size and leached with water under agitation for a long period (usually 10 hours) to obtain a high extraction of the sodium tungstate. The resulting filtrate is acidified with hydrochloric acid to produce tungstic acid precipitate, which, when heated to about 1,000° C. is converted to tungsten trioxide. The reaction is H_2WO_4 + Heat = WO_3 + H_2O.

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Preparation of Pure Oxide

The tungstic acid, oxide, or tungstate prepared by any of the various methods of extracting tungsten from its ores or concentrate is always impure and requires further treatment. The most usual impurities are iron, manganese, calcium, molybdenum, alumina, and silica. Others include lead, copper, titanium, sulfur, phosphorus, arsenic, antimony, bismuth, and tin, depending on the type of tungsten ore treated.

Purification can be accomplished by several methods (33, 41). The simplest method begins with the sodium tungstate that results from the water extraction of the fused ore during decomposition of the ore. The soluble tungstate can be crystallized or treated with calcium to precipitate calcium tungstate, which reacts with hydrochloric acid, yielding tungstic acid. The tungstic acid is dissolved in an excess of ammonium hydroxide, and the impurities are removed by settling or filtration. Reprecipitation with hydrochloric acid and re-solution in ammonium hydroxide yield a high-purity ammonium tungstate, which, when converted to a tungstic acid precipitate and ignited at 1,000° C., results in a high-purity tungsten trioxide product.

Reduction to Metal Powder

Tungsten powder, for use as pure metal, is produced by hydrogen reduction of purified tungsten trioxide. For use in manufacturing tungsten sheets, wire, etc., it must be free of impurities and of proper particle size, usually from 1 micron or less to 10 microns. The purified tungsten trioxide powder is spread in thin layers on the bottom of iron or nickel "boats," then introduced at the cooler end of an electric tube furnace. The tube-furnace initial temperature is about 550° C., and the final temperature is approximately 1,200° C. Variables that affect particle size are time, temperature, and flow of hydrogen through the furnace. The "boats" are gradually pushed through the iron tubes against a stream of hydrogen entering from the opposite direction. Seven hours is usually required for the reduction.

The tungsten trioxide powder in the "boats" goes through several stages of reduction before it is reduced to metal. The WO_3 is yellow, the W_2O_5 deep blue, and WO_2 chocolate brown; they appear in the order named. The equation for the reaction is:

$$WO_3 + 3H_2 = W + 3H_2O.$$

For use in certain tungsten alloys or cast tungsten carbides, some latitude is allowed in the amount of impurities present in the metal powder. As the state of its subdivision is of minor importance, several methods for producing the metal can be used, such as the reduction of tungsten trioxide with carbon in an electric furnace, or in graphite or fire-clay crucibles; reduction with metals such as aluminum, zinc, or magnesium; or electrolytic reduction. The method selected must be simple and economical for production of the desired alloy. A direct reduction of tungsten concentrate with carbon can also be made, but the product produced has not been widely adopted.

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Conversion of Powder to Massive Metal

Tungsten powder is converted into solid metal by compacting in steel dies with a two- or three-way hydraulic press and sintering. An electric current which through the pressed bars provides the heat for sintering.

Before 1907 the tungsten metal produced was hard and brittle and could not be worked by mechanical means. The search for a method to produce tungsten in ductile form, so that it could be drawn into fine wire for electric lamp filaments, disclosed that a very pure tungsten powder is necessary for this purpose. Tungsten was first produced in ductile form (filament wire) through research work by Fink and Coolidge, two General Electric Co. investigators. Their work in making ductile tungsten from the metal powder by suitable heat treatment and mechanical working was the basis for development of the modern process.

In the production of ductile tungsten the high-purity powder is first pressed into bars approximately 24 inches long by 1/2 or 3/4 inch thick at a pressure of about 20 tons per square inch. At this stage the pressed bars are too fragile to handle. Therefore, they are pre-sintered in a hydrogen atmosphere, in an electric furnace, consisting of an alundum muffle wound with a molybdenum heating element, at 1,000° to 1,200° C. for about one-half hour. The bars are then cooled to room temperature in a water-cooled chamber at the rear of the muffle. At this stage the sintered bars are still porous, very brittle, and lack crystalline structure but are strong enough for handling.

The next step is to produce the desired crystalline structure and to remove the voids by sintering with an electric current. A bar is suspended vertically from a water-cooled clamp, while the lower end is clipped to a copper or molybdenum contact immersed in a water-cooled copper hood which dips into a ring of mercury, and dry hydrogen is passed through the enclosure. The electric current is then passed through the bar until its temperature is raised to nearly the melting point (about 3,200° C.). The bar is cooled to room temperature before it is removed from the furnace.

For the production of rod or wire a bar is heated to 1,500° to 1,600° C., making it malleable so that it can be rolled or hammered. The heated bar is hammered in a swaging machine, which gradually reduces the diameter to the desired size. For wire drawing it is swaged until the rod reaches 0.1 inch in diameter, the size at which it can be drawn. The drawing operations are carried out under a gradually reduced temperature as the diameter of the wire decreases.

Tungsten sheets are produced by hot working. The heated bar is first forged, then rolled to the required thickness.

Production of Cemented Tungsten Carbide and Hard-Facing Material

Tungsten carbide is produced in two chemical forms, tungsten carbide (WC) and tungsten subcarbide (W_2C). The former is present in sintered carbides, and both are present in the fused carbides.

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In preparing cemented tungsten carbide, extremely fine tungsten-metal powder of high purity is used. After ball milling with the necessary amount of carbon, usually lampblack, the tungsten powder is carburized to produce tungsten carbide (WC). The amount of carbon used is slightly in excess of the theoretical 6.12 percent in tungsten carbide. The mixture is heated either in a high-frequency crucible furnace or a tube furnace at about 1,500° C. for 1 to 2 hours. After cooling, under hydrogen, the tungsten carbide is crushed and milled (11, pp. 1381-1383); (33, pp. 231-234) to pass a fine screen.

The prepared tungsten carbide is then further milled in a stainless steel ball mill with cobalt powder, which coats each particle of tungsten carbide with a thin pellicle of that metal. The amount of cobalt added depends on the intended application of the final product. Increased cobalt additions increase strength and shock resistance of the product and decrease hardness. After milling, the powder is pressed in molds under hydraulic pressure to form billets of the desired shape. These billets are heated in hydrogen at a temperature (800° to 1,000° C.) that permits enough cohesion to prevent breaking while the material is being shaped with a steel tool to the desired form. Final sintering of the shaped article, packed in carbon or alumina, is accomplished at between 1,400° and 1,500° C. for 30 to 60 minutes in a hydrogen atmosphere. The time and temperature employed depend on the cobalt content of the article. During this last step cementation takes place. The final product consists of particles of tungsten carbide cemented together with cobalt.

There are three important methods for shaping cemented carbides - cold press, hot press, and extrusion. The cold-press method has been described above. The hot-press method is utilized principally for making large pieces and intricate shapes. It differs from the cold-press method in that pressing and sintering are carried out at the same time. Pressure ranging from 500 to 2,500 p.s.i. is applied to the carbide-cobalt mixture while it is being heated to a sintering temperature by a high-frequency current. The pressure is maintained high enough to press the material in final form.

The extrusion method is essentially utilized for making rods, tubes, angles, twist drills, etc. The bonding material used in this method is mixed with a suitable plasticizer, such as sugar dissolved in alcohol. The mass is then extruded under high pressure in the desired shapes. Low-temperature volatilization of the organic binders and sintering treatment are similar to the one used in the cold-press method.

Fused carbides are usually made by melting and casting scrap or tungsten powder with carbon. The castings are more brittle than sintered material and therefore, have a more restricted range of usage. However, they are harder and less costly than sintered tools and superior in certain applications. Items such as chemical spray nozzles, sand-blasting nozzles, valves, and valve seats for gaslines, which contain abrasive particles, can be made by casting. Fused carbides are extensively used in granular form as a filler material in hollow-steel welding rods, which can then be used to produce a hard-wear-resistant surface. The procedure for producing fused carbides is briefly described as follows:

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Technically pure tungsten powder (99.0 to 99.5 percent pure), ranging in particle size from 50 to 500 microns, is mixed with tungsten metal and tungsten carbide scrap. If carbide scrap is not available, about 3 percent of carbon is added in the form of graphite or lampblack. To improve castability and obtain a graphite-free carbide, 5 percent tantalum carbide may be added to the charge, before it is placed in a graphite crucible to which is attached a mold. The entire assembly is then placed in a specially designed horizontal-carbon-tube tilting furnace or in a high-frequency vertical furnace and heated to about 3,000° to 3,250° C. (5,432° to 5,882° F.). When the carbide becomes molten, it is poured in the mold and cooled to about 200° C.; then the mold and crucible are removed from the furnace. When the casting becomes dark red, it is broken away from the mold and cleaned for use as such or crushed to desirable particle size for use as a filler for welding rods.

Production of Ferrotungsten

Ferrotungsten is produced from all types of tungsten ores but generally direct from clean concentrates containing at least 60 percent WO_3. Satisfactory products are obtained with wolframite, scheelite, ferberite, or hübnerite, separately or in combination, provided that the iron-tungsten ratio is such that it will permit formation of an alloy containing 70 percent or more tungsten. A usual combination is wolframite, 70 percent; scheelite, 25 percent; and ferberite, 5 percent. The reduction may be accomplished by several methods - namely (1) carbothermic, (2) aluminothermic, (3) silicothermic, (4) metallothermic, and (5) electric furnace. The electric furnace method is used most in the United States.

In the crucible method ferrotungsten is produced mainly by reducing concentrates or tungstic acid with carbon in crucibles. In this process a charge of concentrates, flux, steel, or scrap iron and coke or charcoal is placed in a clay-lined crucible and heated in a gas-fired furnace.

In the aluminothermic method powdered aluminum is used to reduce tungsten ore or concentrate in an electric furnace. The particle size of the ore and the amount and size of the aluminum added control the speed of reaction and the efficiency of operation. Proper fluxing with line, soda ash, and fluorspar, along with the aluminum, removes the gangue as slag. Since aluminum reduces metallic oxides, only high-purity concentrates are used to alleviate contamination of the charge.

In the silicothermic method ferrosilicon is used; otherwise, the process is the same as the aluminothermic method. Ferrosilicon is used because of the lower cost. In this process the concentrate (generally scheelite) is charged into a molten bath of ferrosilicon in an electric furnace employing ferrosilicon electrodes.

In the metallothermic method (a combination of the aluminothermic and the silicothermic methods) no extraneous application of heat is used. The reaction is started by the chemical reaction of a highly reactive ignition mixture composed of barium peroxide and ferrosilicon in the center of a charge of ore, ferrosilicon, and aluminum. When the reaction temperature is reached (about

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5 minutes), reduction occurs, leaving a button of solidified tungsten in the hearth. A knockdown, hearth-type furnace is used for this type of operation.

In the electric furnace method the ore or concentrate, in admixture with ferrosilicon and basic fluxes, is fed into small arc furnaces of the knockdown type. The slag is tapped at intervals while the metal is being collected as a solid block of alloy or a "button" - the most commonly used term. When the button reaches 8,000 to 14,000 pounds, the furnace is knocked down (dismantled), and the solid button of metal is cleaned and crushed to about 1 inch or finer. Text books by Li (33), Gregg (22, pp. 12-19), and Yntema and Percy (47, pp. 483-500) fully describe the reduction method of ferrotungsten in the modern electric furnace.

Economical production of high-grade ferrotungsten from scheelite, wolframite, and ferberite involves many problems. Copper, arsenic, tin, lead, antimony, bismuth, manganese, and sulfur are troublesome. Furthermore, specifications for carbon and silica in the ferrotungsten must be met (4, p. 391). Preparation of reasonably pure tungsten trioxide by wet methods and electric furnace reduction of this oxide are steps through which a very high grade product may be obtained.

Use in Steel and Other Alloys

In the production of tungsten steel tungsten may be added to the molten-steel bath as scheelite, as ferrotungsten containing 75 to 85 percent tungsten, as tungsten-steel scrap, or as tungsten-metal powder.

In 1956 the total consumption of tungsten in the United States was 42 percent in steel, 35 percent in tungsten carbides, and 13 percent in pure metal used for lamp filaments, switch contacts, or electronics equipment. Consumption of tungsten in 1956, by class of manufacture, is given in table 37.

Tungsten additions of only 1 to 2 percent in tool steels result in tough and fine-grained-structure steels that are less likely to distort or crack upon hardening. Additions of tungsten in the lower percentages (under 6 percent) improve shock resistance; a higher percentage of tungsten additions improves wear resistance and hardness at working temperatures. High-speed steels normally contain 18 percent tungsten, 4 percent chromium, 1 percent vanadium, and 0.5 to 0.8 percent carbon. The chief property of tungsten in steel, in the presence of chromium, is its ability to retain considerable hardness at high temperatures (39, pp. V-2, 3). A carbon content between 0.67 and 0.73 percent gives the best combination of hardness, cutting ability, wear, and shock resistance. The high-carbon types of high-speed steels are used in cutting tools, such as broaches, facing, boring, turning, and threading tools, files, glass cutters, milling cutters, reamers, shear blades, lathe tools, and special carbide saws. When wear resistance is imperative, the high-carbon-type steels are used for manufacturing such articles as ball-mill plugs, Brinell balls, burnishing tools, centers for lathes and grinders, dies for extruding, swaging, wire drawing, gate-contact points, rock drills, sand-blast nozzles, teeth and jaws for excavators, and many other articles.

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TABLE 37. - Consumption of tungsten, by class of manufacture, in 1956

	Thousand pounds of contained tungsten
Steel:	
High-speed	2,893
Other tool steel	522
Alloy steel (other than tool)	531
High-temperature, nonferrous alloys	342
Other nonferrous alloys	56
Metal (wire, rod, and sheet)	1,242
Carbides: Cemented, sintered, other	1/ 3,426
Chemicals:	
Fluorescent powders	54
Pigments	93
Other	6
Miscellaneous 2/	557
Total	9,722

1/ Estimated to be 60 percent cemented or sintered and 40 percent hardfacing and other.

2/ Includes uses (not classified by reporting firms) in diamond-drill bits, electrical contact points, welding rods, etc.

In the ferrous and nonferrous alloy, tungsten is alloyed with other elements, such as chromium, cobalt, nickel, manganese, copper, molybdenum, and silver, to produce alloys for special purposes.

Cobalt-chromium-tungsten alloy is "stellite" used as a cast-cutting tool. It maintains hardness and strength at temperatures above 1,000° F. and is used for hard-faced plates, rolls, crushers, exhaust valve seats, and pump parts subject to erosion and corrosion. It usually contains 38 to 60 percent cobalt, 5 to 20 percent tungsten, and 30 percent chromium. The carbon content may range from 1 to 2.45 percent, and the cobalt content may be lowered appreciably by adding 4 to 5 percent of molybdenum without impairing hot hardness. Sometimes 0.2 percent boron is added to enhance performance (39, pp. V-6).

ANALYSIS OF TUNGSTEN

Detection

For the most part, tungsten minerals are satisfactorily decomposed by the hydrochloric-nitric-acid treatment, provided the material is ground to at least 200-mesh. By this treatment, scheelite and hübnerite are easily decomposed; ferberite is more difficult to decompose and wolframite the most difficult to decompose and wolframite the most difficult of all (27, p. 682).

To detect tungsten minerals in ores chemically, the finely powdered material is heated with aqua regia until the lemon-yellow tungstic acid is

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obtained; then it is treated with hydrochloric acid and zinc, aluminum, or tin is added. If tungsten is present, a blue color forms in the solution or the yellow residue turns blue owing to reduction by the nascent hydrogen (38, p. 148).

To detect very small amounts of tungsten, thoroughly mix one-half gram of the finely powdered material with 4 grams of sodium carbonate, and fuse. Dissolve the fused mass in boiling water, and acidify the aqueous solution with an equal volume of hydrochloric acid, and add a small piece of tin; warm gently, if necessary. A fine blue color of either the solution or the residue indicates the presence of tungsten. A brown color is obtained if reduction is continued long enough. Tin is preferred to zinc for reduction because, if only a small quantity of tungsten is present, the zinc reduces it very quickly to the brown oxide; and the blue color is unnoticed. The action of tin is slower and much more certain.

Columbium is the only other element that gives a blue color followed by a brown color under the conditions of this test. However, the columbium blue is less brilliant, and it can be distinguished from the blue of the tungsten trioxides by the fact that it disappears when the blue solution is diluted with water. Vanadium gives a blue color when solutions of its salts are reduced; but tartaric acid also will cause this reduction. However, it will not reduce tungsten trioxide. Molybdenum on reduction goes through a series of color changes from violet to blue to black. Titanium gives a violet color. No other elements will interfere with the reduction test for tungsten.

The use of the ultraviolet lamp (sometimes referred to as "black light") for detecting tungsten mineral is limited to scheelite, a fluorescent mineral. It is not effective for detecting ferberite, wolframite, or hübnerite, as these minerals do not fluoresce under the ultraviolet light. Scheelite fluoresces bright blue in the dark under the action of ultraviolet light. Use of this light has led to the discovery of important scheelite deposits.

When invisible ultraviolet rays strike a mineral, it causes displacement of an outer electron in the atom, causing emission of fluorescent light having a greater wavelength than the incident ultraviolet ray. Fluorescence of scheelite is commonly blue, sometimes blue-white, cream or golden-yellow, but never green, red, or pink (44).

Quantitative Analysis

Determination in Ores and Concentrates

Since there are no precise volumetric procedures for determining tungsten, the gravimetric methods are preferred. The element is determined as tungsten trioxide, WO_3. It may be isolated in the form of tungstic acid, ammonium tungstate, mercurous tungstate, and preferably by precipitation with cinchonine, all of which yield the trioxide (WO_3) on ignition.

The trioxide (WO_3) may be contaminated with silver, iron, molybdenum, phosphorus, chromium, silica, tin, antimony, columbium, or tantalum compounds.

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The fusion of pure trioxide with sodium carbonate, and extraction with water leave no residue. Sodium, potassium and ammonium slats, phosphorus, arsenic, and molybdenum retard precipitation. A temperature of 750° C. is necessary for dehydration of tungstic acid. The ignition temperature should not exceed 850° C. owing to the volatility of WO_3. In the ordinary course of analysis tungsten tends to separate with silica and vice versa, when the silica present is large. For this reason, it is highly important that, when tungsten is determined in siliceous ores, the tungsten be separated from the silica by volatilizing the silica with hydrofluoric acid alone or in combination with sulfuric acid in a platinum crucible.

The following procedure for determining tungsten in ores and concentrates is a modified gravimetric method utilizing digestion with acids and precipitation with cinchonine that has been used for many years at the Reno Station of the Bureau of Mines on various tungsten ores with highly satisfactory results. This modified method is based on the assumption, proved by laboratory experience, that by far the greater majority of tungsten samples analyzed do not contain the interfering elements, such as tin, columbium, tantalum, and titanium. Should ores contain these impurities, they must be removed. The impurities in scheelite ores often are silica, pyrite, and sometimes molybdic oxide or sulfides. A detailed description of the general method for separating these elements is contained in the textbooks and bulletin by Hildebrand (27, p. 682), Scott (40, p. 1005), and Moore (35, p. 131).

Procedure

Ore samples are prepared by grinding in a laboratory Braun pulverizer to pass through a 200-mesh screen. Previously, the samples were ground in an agate mortar, but several check analyses of duplicate samples ground in the pulverizer and in the agate mortar to pass a 200-mesh screen showed no difference in assay results. The pulverizer is utilized because of speed in preparing a large number of samples daily.

A 5-gram sample is taken for analysis for most ores. Smaller amounts are taken for high-grade ores and 0.5- to 1.0-gram samples for high-grade concentrates.

Place the weighed 5-gram sample in a 400-ml. beaker, digest with 200 ml. of concentrated hydrochloric acid, and cover. Boil gently until the volume is reduced to 25-50 ml. At this point start precipitation of the tungstic acid by adding 5-10 ml. of concentrated nitric acid, and continue boiling until the volume is reduced to about 5 ml. Add 200-250 ml. of hot water, followed by 6-8 ml. of cinchonine solution (a)[5/] to complete precipitation of the tungsten. Allow the sample to stand at least 4 hours, preferably overnight. Filter the resulting precipitate of tungstic acid, silica, etc. Wash the precipitate five times with cinchonine wash solution (b), then with one cold-water wash. Then rinse the washed precipitate back into the original beaker with hot ammonia solution (c), and add 6 ml. of concentrated ammonia

5/ Letters in parentheses refer to items in Appendix I at the end of this report.

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hydroxide. Heat the contents of the covered beaker to boiling until all the yellow tungstic acid is dissolved. Then filter the solution through the same filter paper used in the initial filtration, and wash the precipitate six times with hot ammonia wash solution (c). Discard the washed precipitate, and boil the resulting ammoniacal filtrate containing the tungsten quickly to expel all traces of ammonia. Dilute the filtrate to 200-250 ml. with hot water, and 5 ml. of concentrated hydrochloric acid and 6-8 ml. of cinchonine solution (a) to reprecipitate the tungstic acid. Stir the solution vigorously to coagulate the precipitate. Allow the solution to stand at least 4 hours, preferably overnight; then filter and wash the reprecipitated tungstic acid five times with cinchonine wash solution (b). Transfer the filter paper and precipitate to a platinum dish or crucible, and heat them on a hot plate until moisture is expelled and charring of the filter paper begins; then place the dish or crucible in an electric muffle furnace and ignite it at 800° C. until all the carbon has been burned off and the tungstic acid converted to the oxide, WO_3. At this point the WO_3 is not pure; the principal contaminating elements are silica, iron, and usually molybdenum.

Remove the silica by treating the igniting WO_3 in the platinum dish or crucible with a few milliliters of hydrofluoric acid and evaporating to dryness on a hot plate, followed by a short ignition in the electric muffle furnace at 800° C. After removal of the silica, cool and weigh the platinum dish or crucible containing the WO_3, together with small amounts of iron and possibly molybdenum. Add enough sodium carbonate to produce a fluid melt to the weighed platinum dish or crucible (usually 1 or 2 grams is adequate for most ore samples, but more is required for precipitates from concentrates and other high-grade material)

Next fuse the contents of the dish or crucible over the flames of a Fischer or Meker-type burner, until a clear liquid melt is obtained. When cool and solid, extract the fused mass with boiling water, washing the solution through a filter that retains any $Fe(OH)_3$ precipitate formed; the tungsten passes into the filtrate as Na_2WO_4. Wash the $Fe(OH)_3$ precipitate several times with hot water to remove all of the Na_2WO_4 and Na_2MoO_4, if molybdenum is present. Then transfer the washed $Fe(OH)_3$ precipitate and filter paper back to the original washed platinum dish or crucible, dry, and ignite in an electric muffle furnace at 800° C. until the carbon is burned off and the $Fe(OH)_3$ is converted to Fe_2O_3. Cool and weigh the crucible or dish and residue. This weight, subtracted from the first weight obtained, gives the weight of WO_3 plus MoO_3 from any molybdenum present.

Molybdenum is corrected by colorimetric determination in the filtrate from hot-water extraction of the sodium carbonate fusion. The colorimetric determination is made by the $SnCl_2$-KSCN method after the Na_2CO_3-Na_2WoO_4 filtrate is acidified with sulfuric acid, using tartaric acid to prevent precipitation of tungstic acid. The amount of molybdenum determined colorimetrically is calculated to MoO_3 and deducted from the weight of impure WO_3 determined by the two weighings; the difference is reported as WO_3.

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Colorimetric Determination of Molybdenum

The alkaline filtrate obtained above contains the tungsten and molybdenum; not more than 100 ml. in volume is transferred to a 250-ml. separatory funnel. One gram of tartaric acid is added and the solution made weakly acidic with 1-1 sulfuric acid and cooled to 15° C. Twenty-five ml. of 8-percent ferric sulfate solution (d) and 10 ml. of 5-percent KCNS solution are added. The stoppered flask and contents are shaken vigorously for several minutes, then treated with 5 to 10 ml. of stannous chloride solution (f) and again shaken vigorously for several minutes. Tungsten may precipitate upon addition of ferric sulfate, but it does not interfere.

The stannous chloride solution reduces the iron from the ferrous to the ferric state and the molybdenum from the sexivalent to the quinque-valent or quadrivalent condition. The potassium thiocyanate reacts with the reduced molybdenum to give an amber or reddish-brown color to the solution, depending on the amount of molybdenum present. Next, the solution is cooled to 15° C., 50 ml. of ether is added, and the separatory funnel is stoppered and shaken vigorously several minutes and allowed to stand until the liquid has separated into two distinct layers. The acid layer, which will contain all of the tungsten, is drawn off and discarded (at this point, if the lower layer shows any pink color, it must be given a second treatment with ether), and the upper or ethereal layer will contain all of the molybdenum. It is drawn off into a 50-ml. comparison tube and diluted with ether to the 50-ml. mark. The comparison tube is stoppered with a soft cork to prevent evaporation of the ether, and its contents are mixed thoroughly by manipulating the tube in the usual manner. After standing for several minutes it is ready for comparison with the standard. The weight of the molybdenum in the sample is then determined by comparing the intensity of the color of the ether solution of potassium molybdenum thiocyanate with the standard solution that contains a known amount of molybdenum. Any molybdenum found is calculated to MoO_3 and deducted. The weight of the impure WO_3 precipitate, minus the weight of Ta_2O_5 + Cb_2O_5, Fe_2O_3, and MoO_3 found, gives the weight of WO_3 in the sample. The weight of WO_3 is multiplied by 79.31 and divided by the weight of the sample taken to give the percentage of tungsten.

The standard for comparison is prepared by placing 25 ml. of 8-percent ferric sulfate solution (d) in a 250-ml. separatory funnel and adding the standard molybdenum solution (g) from a burette. The solution is diluted with cold water to approximately 100 ml., and development of the molybdenum color and extraction is completed as above for the sample. It is advisable to allow the molybdenum solution to stand in the comparison tube several minutes before comparing it with the standard, as the intensity of the color sometimes changes at first but remains stable thereafter for several days, and even longer if kept in the dark when not in use.

The percentage of molybdenum in the sample is then determined by comparing the intensity of the color of the ether solution of potassium-molybdenum-thiocyanate with that of the standard. The comparison can be made by use of a colorimeter or spectrophotometer and preparation of a calibration curve. An alternate method follows: The darker of the two solutions (the sample and the

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standard) is diluted carefully with ether and mixed thoroughly until they match exactly. The amount of molybdenum per milliliter in the standard is then figured and the weight of molybdenum in the precipitate is calculated by multiplying the weight of molybdenum in each milliliter by the number of milliliters. The following example illustrates the calculation: In comparing the standard with the sample, 3 ml. of standard molybdenum solution (1 ml. = 0.0002 gm. Mo) was used and diluted to 44 ml. with ether, therefore: 3 x 0.0002 = 0.0006 gm. Mo; or 0.0006/44 = 0.0000136 gm. Mo per ml. The sample was diluted to 36 ml.; hence 36 x 0.0000136 = 0.00049 gm. Mo in the sample, 0.00049 gm. Mo x 1.5 = 0.000735 gm. MoO_3 in the sample.

Colorimetric Determination of Tungsten in Low-Grade Ores

The following short colorimetric method is a modification of the method developed by Grimaldi and North (23). It is used by the Nevada Scheelite Corp. for estimating the WO_3 content of low-grade mine ores and mill feeds.

Procedure

Fuse 1 gram of pulverized sample with 5 grams of sodium peroxide in a 50-ml. Armco-iron crucible with cover (a Braun-iron crucible No. 23935 and a Merck blast burner). The fusion requires about 4 minutes. Cool and tap the melt into a dry, 400-ml. beaker. Add 100 ml. of boiling alcohol wash solution (h). Wash the crucible and cover into the beaker. Boil approximately 5 minutes to eliminate the excess peroxide. Cool, and transfer to a 250-ml. volumetric flask, diluting to the mark with 1-percent sodium hydroxide solution. Allow to settle, and shake. Filter on a dry Whatman No. 40 paper into a dry beaker until about 40 ml. of filtrate has been collected. Pipette a 25-ml. portion of this filtrate into a 100-ml. volumetric flask. Cool with tap water, add 10 ml. of concentrated sulfuric acid very slowly, and cool. Add 5 ml. of $FeSO_4$ solution (i), 20 ml. of concentrated HCl, and 5 ml. $SnCl_2$ solution (j). Place the flask in a boiling-water bath for 5 minutes, then cool in a running-water bath to room temperature. Add 10 ml. KSCN solution (k). Cool to 20° C., and dilute to the mark with distilled water. Allow 5 minutes for color development, transfer a suitable portion to a colorimetric tube, and determine the percent transmittance, using a 400-millimicron filter (No. 42 blue colorimeter filter); use a reagent blank carried through the same procedure for a 100-percent transmittance setting. Also, use a sample of standard solution (l) carried through from the point where 25 ml. of the filtrate is pipetted into a 100-ml. volumetric flask, 10 ml. of standard solution, and 25 ml. of water. The standard solution usually gives a reading of 0.81 to 0.86. When the percentage of transmittance shows the tungsten trioxide content to exceed 1.25 percent, a determination can be made at the expense of accuracy by using a corresponding smaller portion of the filtrate and multiplying results by a conversion factor. For example, if 10 ml. is taken, multiply by 2.5.

$$\text{Calculation: } \frac{\text{percent, } WO_3 \text{ unknown}}{\text{Reading of X}} = \frac{0.050 \text{ percent } WO_3}{\text{Reading of standard}}$$

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Rapid Fluorescence Estimation of Molybdenum in Scheelite Ores and Concentrates

Many scheelite ores contain powellite $(Ca, Mo, W)O_4$. In the past the molybdenum content of scheelite tungsten ore could be effectively determined only by chemical analysis. An investigation in the laboratories of the Federal Geological Survey resulted in improvement of the fluorochemical method to such a degree that it may now be used to determine accurately the molybdenum content of commercial scheelite. Pure scheelite fluoresces a characteristic blue. Small amounts of molybdenum alter the emission from blue to white to yellow. When the fluorescence is white, the scheelite contains roughly 0.35 to 1.0 percent molybdenum; when the scheelite fluoresces a distinct yellow, the ore contains more than 1.0 percent. This method is particularly useful for a quick check on a scheelite concentrate to determine if the molybdenum content is subject to a price penalty. The effect of varying proportions of molybdenum and tungsten is shown below:

Mineral	Mo, percent	Fluorescence
Scheelite.....	Negligible	Strong blue
Do.....	Trace	Paler blue
Do.....	0.35-1.0	Strong blue
Do.....	<1.0	Yellow
Do.....	4.8	Strong yellow
Powellite.....	-	Do.
Cuproscheelite.....	-	Yellow

Procedure

Prepare a series of standards, using pure calcium tungstate. Thoroughly mix portions of pure calcium molybdate into the calcium tungstate, so that the standards will contain 0.05, 0.10, 0.15, 0.20, etc., percent molybdenum, the molybdenum content of the series of standards to be within a range best suited for the purpose. Mount the standards on comparison cards; or, place the standards in tin or stainless steel cups 1 inch in diameter and 1/4 inch deep; fill level full. Arrange the standards in a circle; next place the finely ground sample of ore or concentrates in the center of the circle. Then take a card, and cut two holes of the same size into it. Place the card over the sample and standards, with the center hole of the card over the sample and the outer hole over the standards. Hold an ultraviolet light over the card, and rotate the card over the standards until the fluorescence color of the sample matches the fluorescence color of the standard. The accuracy of these determinations can be increased by using a greater number of standards.

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APPENDIX I

Solutions required for tungsten analysis:

a. Cinchonine solution: 125 g. of cinchonine dissolved in 1 l. of 1:1 HCl.

b. Cinchonine wash solution: 30 ml. of cinchonine solution, 30 ml. of concentrated HCl in 1 l. of H_2O.

c. Ammonia wash solution: 100 ml. of concentrated NH_4OH, 5 g. NH_4Cl, 900 ml. H_2O.

d. Ferric sulfate solution: 8%: 80 g. of ferric sulfate is dissolved in 1 l. of 20 percent sulfuric acid.

e. Potassium thiocyanate (KCNS) 5% solution: Dissolve 5 g. of KCNS in 100 ml. of H_2O.

f. Stannous chloride solution: 350 g. $SnCl_2$ is added to 200 ml. of 1:1 hydrochloric acid in a 500-ml. Erlenmeyer flask and the liquid boiled gently until the salt has almost dissolved. Transfer to a liter bottle, and dilute with freshly boiled water to 1,000 ml. Add a few pieces of metallic tin to prevent oxidation.

g. Standard molybdenum solution: 1 ml. equals 0.0002 g. of Mo. Prepare solution by dissolving 0.430 g. of PURE sodium molybdate in 1 l. of water containing 10 ml. of 1:1 sulfuric acid, and mix thoroughly. 100 ml. of this solution is measured carefully with a calibrated pipette into a 250-ml. beaker. 12 ml. of 1:1 sulfuric acid is added, and the solution is put through a Jones reductor into a solution of ferric sulfate and titrated with a standard solution 0.05 N in potassium permanganate (1 ml. equals 0.0016 g. Mo). A blank determination is run on the reductor and the solution of ferric sulfate (this amounts to 0.3 ml.), passing 100 ml. of 6 percent sulfuric acid and 150 ml. of water (the same amount as in the analysis) through the reductor in exactly the same way as the molybdenum solution and titrating the liquid with permanganate. The amount of 0.05 N permanganate required to impart a pink tint to the liquid constitutes the blank to be deducted from the burette reading when the molybdenum solution is standardized.

h. Alcohol wash solution: Add 50 ml. of alcohol (Sheallacol) to 1 l. of distilled water.

i. Ferrous sulfate solution: Dissolve 10 g. of ferrous sulfate in 100 ml. of 10-percent sulfuric acid solution.

j. Stannous chloride solution: Dissolve 23 g. of $SnCl_2$ in concentrated hydrochloric acid, and dilute to 50 ml. with concentrated hydrochloric acid. Prepare fresh daily.

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k. Potassium thiocyanate solution: Dissolve 20 g. of KCNS in 80 ml. of water. Prepare fresh daily.

l. Standard solution: Dissolve 100 mg. of pure ignited WO_3 (or equivalent sodium tungstate) in a few ml. of water that contains 1 g. of N_aOH; heat until clear, and dilute to 100 ml. in a volumetric flask (check the tungstic content gravimetrically). Transfer 50 ml. of the solution to a 1,000-ml. volumetric flask, add 48 ml. of the 50-percent NaOH solution, and dilute to the mark. This is the working standard - 1 ml. equals 0.0500 mg. WO_3.

m. Sodium hydroxide solution, 50 percent: Dissolve 250 g. of N_aOH in water, and dilute to 500 ml.

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BIBLIOGRAPHY

(Arranged alphabetically)

1. AMERICAN CYANAMID CO. Ore-Dressing Notes. No. 7, 2d ed., January 1940, p. 22.

2. BALKE, C. W. Metals of the Tungsten and Tantalum Group. Ind. Eng. Chem., vol. 21, 1929, pp. 1002-1007.

3. BATES, WILLIAM A. Recovery of Scheelite From Ores. U. S. Patent 2,607,479, 1952; Chem. Abs., vol. 46, p. 6055F.

4. BECKET, FREDERICK M. The Development of Ferro-Alloys of Tungsten and Vanadium. Chem. Met. Eng., vol. 30, 1924, p. 391.

5. BONARDI, J. P., AND WILLIAMS, J. C. Treatment of the Tungsten Ores of Boulder County, Colo. Bureau of Mines Bull. 187, 1921, p. 17.

6. BURWELL, BLAIR T. Synthetic Scheelite From Salt Lake Tungsten. Min. World, vol. 17, June 1955, pp. 44-49.

7. BUTLER, B. S., AND VANDERWILT, J. W. The Climax Molybdenum Deposits, Colorado. Geol. Survey Bull. 846-C, 1933.

8. CLEMMER, J. BRUCE, AND O'MEARA, R. G. Flotation and Depression of Nonsulfides, Calcite, Silica, and Silicates, Fluorspar, Barite, Apatite, and Tungsten Minerals. Bureau of Mines Rept. of Investigations 3239, 1934, p. 15.

9. COGHILL, WILL H., AND CLEMMER, J. BRUCE. Soap Flotation of the Nonsulfides. Trans. AIME, vol. 112, 1934, p. 459.

10. COGHILL, WILL H., AND GREEMAN, W. O. Flotation of Fluorspar Ores for Acid Spars. Bureau of Mines Rept. of Investigations 2877, 1938, p. 2.

11. COMSTOCK, GREGORY. Tungsten Carbide - the First Product of a New Metal. Iron Age, vol. 126, 1930, pp. 1381-1383.

12. DANA, J. D., AND DANA, E. S. A System of Mineralogy. J. Wiley & Sons, New York, N. Y., 6th ed., 1911, p. 991.

13. DEAN, R. S., AND DAVIS, C. W. Magnetic Separation of Ores. Bureau of Mines Bull. 425, 1941, p. 214.

14. DEAN, R. S., AND HERSBERGER, A. B. New Flotation Reagents. AIME Tech. Pub. 605, 1935, p. 7.

15. DEVANEY, F. D., AND COGHILL, WILL H. Preliminary Ore-Dressing Tests to Recover Manganese in Rhodochrosite Ores. Bureau of Mines Rept. of Investigations 2902, 1928, p. 2.

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16. DEVANEY, F. D., AND CLEMMER, J. B. Floating of Carbonate and Oxide Manganese Ores. Eng. Min. Jour., vol. 128, 1919, pp. 506-508.

17. DICE, C. M. Methods and Costs of Concentrating Hübnerite Ores at the Ima Tungsten Mine, Lemhi County, Idaho. Bureau of Mines Information Circ. 7230, 1943.

18. ESPENSHADE, G. H. Tungsten Deposits, Vance County, N. C., and Mecklenburg County, Va. Geol. Survey Bull. 948-A, 1947, 17 pp.

19. FERRAND, JEAN. Concentration of Grains of Tungsten Minerals of Less Than 75 Microns on Vibrating Tables of French Design. Rev. ind. min., vol. 35, 1954, pp. 38-68.

20. GANDRUD, B. W., AND DEVANEY, F. D. Preliminary Examination of Low-Grade Bauxite, With Particular Reference to Flotation. Bureau of Mines Rept. of Investigations 2906, 1928, p. 3.

21. GAUDIN, A. M., AND MALEZEMOFF, PLATO. Recovery by Flotation of Mineral Particles of Colloidal Size. Jour. Phys. Chem., vol. 37, 1933, pp. 597-607.

22. GREGG, J. L. The Alloys of Iron and Tungsten. Engineering Foundation, McGraw-Hill Book Co., Inc., New York, N. Y., 1934, pp. 12-19.

23. GRIMALDI, F. S., AND NORTH, VICTOR. Colorimetric Determination of Tungsten in Low-Grade Ores. Geol. Survey Press Release 25900.

24. HESS, F. L. Tungsten Minerals and Deposits. Geol. Survey Bull. 652, 1917, pp. 37-45.

25. HESS, F. L., AND LARSON, E. S. Contact-Metamorphic Deposits of the United States, Geol. Survey Bull. 725-D, 1922, p. 249.

26. HESS, F. L., AND SCHALLER, W. T. Colorado Ferberite and the Wolframite Series. Geol. Survey Bull. 583, 1914.

27. HILDEBRAND, W. F., LINDELL, G. E. F., HOFFMAN, J. T., AND BRIGHT, H. A. Applied Inorganic Analysis. 2d ed., John Wiley & Sons, New York, N. Y., 1953, p. 682.

28. HOBBS, W. H. The Old Tungsten Mine at Trumbull, Conn. Geol. Survey 22d Annual Report, pt. 2, 1901, pp. 13-22.

29. LAWRENCE, H. M., AND DEVANEY, F. D. Flotation of Low-Grade Phosphate Ores. Bureau of Mines Rept. of Investigations 2860, 1928, p. 1.

30. LAWRENCE, H. M., AND ROCA, E. Flotation of Low-Grade Phosphate Ores, II, Bureau of Mines Rept. of Investigations 3105, 1931, p. 1.

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31. LEAVER, E. S., AND ROYER, M. B. Flotation for the Recovery of Scheelite From Slimes Material. Bureau of Mines Tech. Paper 585, 1938.

32. LEE, OSCAR. Flotation of Limestone From Siliceous Gangue. Bureau of Mines Rept. of Investigations 2744, 1926, p. 2.

33. LI, K. C., AND WANG, C. Y. Tungsten. Am. Chem. Soc. Mon. 94, 3d ed., Reinhold Pub. Co., New York, N. Y., 1955, 505 pp.

34. LOVERING, T. S. Tungsten Deposits of Boulder County, Colo. Geol. Survey Bull. 922-F, 1940.

35. MOORE, R. B. Analytical Methods for Certain Metals. Bureau of Mines Bull. 212, 1923, p. 131.

36. OSBORN, W. X. Tabling Tungsten Ores Without Water. Eng. Min. Jour., vol. 123, 1927, pp. 287-289.

37. PEARSON, A. Sink-and-Float at Halkyn Mill. Inst. Min. Met. (London), June 15, 1939, p. 7.

38. RUNNER, J. J., AND HARTMAN, M. L. The Occurrence, Chemistry, Metallurgy, and Use of Tungsten. S. Dak. Sch. of Mines Bull. 12, 1918, p. 148.

39. SACHAROV, PAUL, LEMMON, DWIGHT M., AND ROSS, DONALD C. Materials Survey - Tungsten. Business and Defense Services Administration, U. S. Dept. of Commerce, December 1956, 115 pp.

40. SCOTT, W. W. Standard Methods of Chemical Analysis. 2d ed., vol. 1, 1939, p. 1005.

41. SMITHELLS, C. J. Tungsten. 3d ed., Chapman & Hall, London, 1952, 326 pp.

42. SWAINSON, S. J. Progress in Flotation Practice and Equipment. Eng. Min. Jour., vol. 131, 1931, p. 69.

43. THIELEMANN, R. H. Are We Overlooking Tungsten? An address before American Institute of Mining, Metallurgical and Petroleum Engineer, Convention in New Orleans, La., Feb. 26, 1957.

44. ULTRA-VIOLET PRODUCTS, INC. Rocks and Minerals That Fluoresce With Mineral Light. 145 Pasadena Ave., South Pasadena, Calif.

45. VANDERBURG, WILLIAM O. Tungsten, Pt. I. Bureau of Mines Inf. Circ. 6821, 1935.

46. _____. Mining and Milling Tungsten Ores. Bureau of Mines Information Circ. 6852, 1935, p. 4.

47. YNTEMA, L. F., AND PERCY, ALLAN L. Tungsten. Rate Metals Handbook, edited by Clifford A. Hampel, Reinhold Pub. Co., 1954, pp. 483-500.

48. YOUNG, GEORGE J. Experiments in Magnetizing Roasting. Eng. Mining Jour., vol. 108, 1919, pp. 176-177.

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